

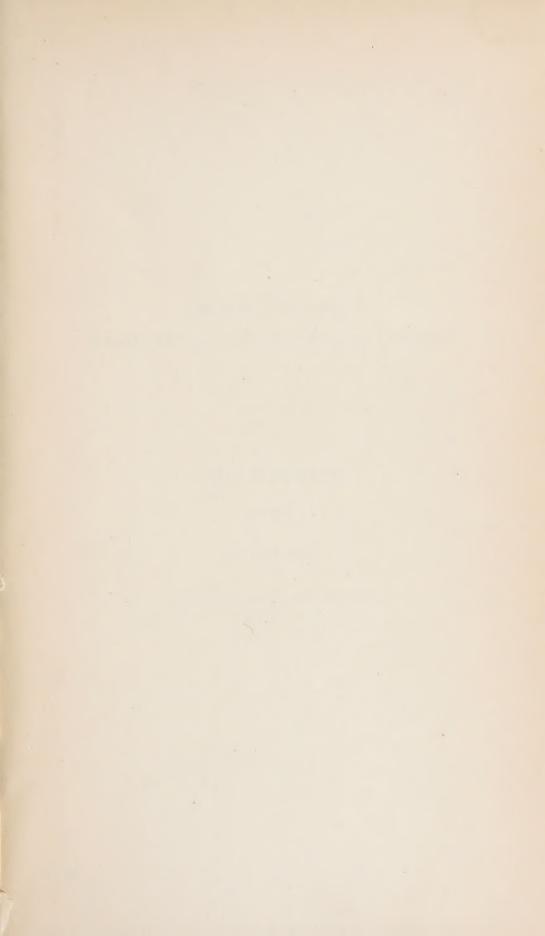
PROCEDENCS

PART II. MINERO

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Second (*Triennial*) Empire Mining and Metallurgical Congress

PROCEEDINGS

PART II

MINING

(SECTION A OF CONGRESS)

PROCEEDINGS OF THE SECOND (TRIENNIAL) EMPIRE MINING AND METALLURGICAL CONGRESS, CANADA, 1927.

Part I.—Report of Proceedings, Inaugural and other Addresses, and General Papers.

Part II.-Mining.

Part III.—Petroleum.

Part IV.—Iron and Steel.

Part V.—Non-Ferrous Metallurgy.

The price of these Proceedings is \$10.00 for the whole of the five parts, or any part may be obtained separately for \$3.00.

Second (*Triennial*) Empire Mining and Metallurgical Congress Held in Canada, August 22nd to September 28th, 1927.

PROCEEDINGS

Edited by

R. P. D. GRAHAM

PART II

MINING

(SECTION A OF CONGRESS)

MONTREAL, CANADA:
PUBLISHED AT THE OFFICES OF THE CONGRESS
923 DRUMMOND BUILDING, MONTREAL.

1928

/110 .E6 1921 pt.2

Second (*Triennial*) Empire Mining and Metallurgical Congress

Canada, August 22nd to September 28th, 1927

Convening Body

THE CANADIAN INSTITUTE OF MINING AND METALLURGY.

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Second (*Triennial*) Empire Mining and Metallurgical Congress

PREFACE

The conditions under which the Second Congress was held were different in several notable respects from those of the First Congress, which met in London in June, 1924. That meeting, convened by institutions and associations representing the mineral and metal industries, the colliery proprietors, and the iron and steel manufacturers, in the United Kingdom, lasted only four days, and the sessions were all held at the British Empire Exhibition, Wembley. It was, of necessity, inaugural in character, and apart from the presentation of a series of noteworthy papers dealing with various phases of the mining and metallurgical industries of the Empire, it had as its chief business the drafting and adoption of a Constitution and Rules for an Empire Council of Mining and Metallurgical Institutions. The Constitution as adopted is as follows:

CONSTITUTION

The Council shall consist, until otherwise determined, of two representatives from each of the following Constituent Institutions:

The Institution of Mining and Metallurgy

The Institution of Mining Engineers

The Institution of Petroleum Technologists

The Canadian Institute of Mining and Metallurgy
The Australasian Institute of Mining and Metallurgy

The Chemical, Metallurgical and Mining Society of South Africa

The South African Institution of Engineers

The Mining and Geological Institute of India

The Iron and Steel Institute

The Institute of Metals

OBJECTS OF THE COUNCIL

To serve as an organ of intercommunication and co-operation between the Constituent Bodies and for the promotion and protection of their common interests.

FUNCTIONS OF THE COUNCIL

(a) To foster and maintain throughout the Empire a high level of technical efficiency and professional status.

(b) To convene successive Mining and Metallurgical Congresses within the Empire.

POWERS OF THE COUNCIL

The Council shall have power to initiate projects for consideration with a view to action and generally to direct the affairs committed to it by the Constituent Institutions, but it shall not commit them to any act or financial obligation without first obtaining the sanction of the respective Councils. Further, the Council shall not have any authority, implied or explicit, to interfere in the domestic affairs of any of the Constituent Institutions, whose complete autonomy shall remain intact.

Dr. R. C. Wallace, as President of the Canadian Institute of Mining and Metallurgy at that time, was one of the official delegates of the Canadian Institute attending the First Empire Congress in London, and on his return to Canada he submitted a proposal to the Institute Council that they should consider the feasibility of having the Second Congress meet in Canada. The proposal was received with enthusiasm, and a preliminary canvass of the situation having made it evident that the Institute might safely assume the responsibility of acting as convenor, an invitation was forwarded to the Empire Council to hold the Second Congress in Canada in 1927. This invitation was accepted by the Empire Council on behalf of the Constituent Institutions.

The Second Congress opened in Montreal on August 22nd, with addresses by the Honorary President, The Right Hon. Sir Robert S. Horne, G.B.E., K.C., M.P., and the President, the Hon. Charles Stewart, M.P., Minister of Mines for Canada. Technical sessions were held on this and the following day, the proceedings including the presentation of a notable paper by Sir Thomas H. Holland on a Proposed Review of the Mineral Resources of the Empire. The proposals outlined in this paper were warmly endorsed in addresses by official representatives from various parts of the Empire, and at the conclusion of the discussion the following resolution was adopted:

This Second (Triennial) Empire Mining and Metallurgical Congress, assembled in Canada, having discussed a proposal for instituting a review of the mineral resources and industries in each appropriate administrative unit throughout the Empire, and of the conditions affecting their development, embodied in a paper submitted under the auspices of The Institution of Mining and Metallurgy by Sir Thomas H. Holland,

RESOLVED, that the proposal be referred to the Empire Council of Mining and Metallurgical Institutions, to be transmitted to the Councils of the constituent bodies for consideration, with a request that they will formulate their views and communicate them to the Empire Council for further action.

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On the following day, special trains conveyed the party to Ottawa, where members had the opportunity of meeting His Excellency the Viscount Willingdon, Governor General of Canada, and The Hon. Charles Stewart, representing the Prime Minister, at a luncheon tendered by the Congress.

Following this, two days were devoted to technical sessions in Toronto, and over the week-end there were excursions to Hamilton, Niagara Falls, and other points of interest.

From Toronto, the Congress trains proceeded to some of Ontario's world-famous mining districts, visits being paid in turn to nickel-copper mines at Sudbury, silver mines at Cobalt, and gold mines at Kirkland Lake and Porcupine.

After leaving the Porcupine area, the party divided, those who had elected to take Tour A going westward to the Pacific coast, while Tour B proceeded eastward across Quebec and the Maritime Provinces to Newfoundland. In each case the principal mining centres and metallurgical plants, as well as some hydro-electric plants, were visited, and a number of technical sessions were held: on Tour A, in Winnipeg, Vancouver, Jasper, and Quebec; and on Tour B in Quebec and St. John's, Newfoundland. Tour A covered approximately 7,750 miles, and Tour B 5,525 miles. The final session of the Congress was held in Quebec on September 26th. It is with considerable gratification that we are able to record that, from the opening day to the closing of the Congress, there was not a single case of illness nor an accident of any kind.

Official banquets were held in Montreal, Toronto, Winnipeg, Vancouver, and Jasper, and in St. John's, Newfoundland. At numerous other points the Congress parties were lavishly entertained by mining companies and individuals; and golf and other clubs in the various cities visited were most hospitable in extending facilities for recreation. The Congress is especially indebted to the Ladies' Committees, which, at each stopping place in the course of the tours, were indefatiguable in providing entertainment for the visiting ladies.

Lists compiled from the official registers and other available sources show that the total attendance at the Congress was well over twelve hundred. Of these, 350 participated in

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Tour A and 96 in Tour B. The gathering may be truly described as exceptionally well representative of every important phase of the mining and metallurgical industries in all parts of the Empire. The personnel included official delegates from each of the following institutions:

The Institute of Metals (London)

The Institution of Mining Engineers (London)

The Institution of Mining and Metallurgy (London)

The Iron and Steel Institute (London)

The Institution of Petroleum Technologists (London)

The Chemical, Metallurgical, and Mining Society of South Africa

The South African Institution of Engineers

The Rhodesia Chamber of Mines

The Australasian Institute of Mining and Metallurgy

The Mining and Geological Institute of India

The Canadian Institute of Mining and Metallurgy

In addition, the Congress had the privilege of welcoming official delegates from the United States Bureau of Mines and the American Institute of Mining and Metallurgical Engineers.

A total of 42 papers were presented at the several technical sessions, and with the exception of three these were all printed as 'separates' in advance of the opening of the Congress and distributed to delegates before the meetings. In addition, a 270-page Official Programme was issued giving complete information regarding arrangements for the Congress, with itineraries of the excursions and brief descriptions of the various points of interest visited; and a series of 'short focus' papers was prepared, each giving in concise form particulars of operations at individual mines and metallurgical plants inspected during the tours. Besides these purely Congress publications, specially prepared booklets descriptive of the Dominion's mining and metallurgical industries were issued by the Federal Department of Mines, by the Ontario and Quebec departments of mines, and by several of the mining companies.

The papers submitted to the Congress dealt with a wide variety of topics relating to recent progress, and the present status and future prospects, of the mining and metallurgical industries in almost every section of the Empire where these industries are carried on. Many of the papers have been given wider publicity than was possible at the time of the

PREFACE ix

Congress by publication in the *Transactions* of the constituent institutions, and some also in technical journals. Following the procedure of the First Congress, the complete *Proceedings* are now issued in five Volumes, a full list of the Papers being printed in each volume.

The funds required for carrying out the extensive programme of the Congress were in the main contributed by the Dominion Government, by the governments of the several Provinces, by the Government of Newfoundland, by Canadian mining and smelting companies, and manufacturers of mining machinery, explosives, etc., and by the Canadian Pacific and the Canadian National railways. Generous donations were also received from co-operating Institutions in the United Kingdom, South Africa, Australasia, and India. In all, sixty-six contributions were received, totalling about \$110,000.

The Second Congress was exceedingly fortunate in having, as its Honorary President, the Right Hon. Sir Robert S. Horne, G.B.E., K.C., M.P., who took a most active interest in the proceedings and participated in the Montreal meeting. Much of the success of the Congress was due also to the President, The Hon. Charles Stewart, Minister of Mines for Canada, who rendered great services, both personally and through the Department of Mines, technical officers of which were detailed to accompany each of the special trains, and were available at all times to give information regarding the Dominion's mineral resources and industries. The Congress also received whole-hearted support from the departments of mines of each of the Provinces of Canada, and from the Government of Newfoundland.

Mining companies throughout Canada, and also the two great transcontinental railway systems, the Canadian Pacific and the Canadian National, and the Temiskaming and Northern Ontario railway, took a very active interest in the work of the organization and showed a keen appreciation of its importance to the Empire. The Congress owes much to them for their invaluable co-operation, and also to the management of the various mines and plants visited for their unfailing readiness to do all in their power to make the tours both instructive and enjoyable.

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Acknowledgment must also be made of the notable service rendered by the numerous individuals who composed the various committees responsible for the organization and carrying out of the Congress. The smoothness with which the entire Congress programme functioned is a splendid tribute to the efficient manner in which each of these committees carried out the particular duties assigned to it.

We desire further to record our thanks to the Empire Council of Mining and Metallurgical Institutions, and more especially to its joint general secretaries, Mr. Charles McDermid and Mr. George C. Lloyd, for much helpful advice both during the organization period and while the Congress was in progress. Thanks are also due to the Cunard Steamship Company for the excellent arrangements they made for the comfort of delegates who sailed from England to Canada in the official Congress boat, the *Alaunia*.

Although the Empire Council Banquet, held in London on November 22nd, was an aftermath, and not a part of the actual proceedings, of the Congress, it may be fittingly referred to here since a large proportion of those present had been with the Congress in Canada, and one of its objects was to review the results of the Congress. Notable speeches were delivered by Sir Thomas H. Holland, Chairman of the Empire Council, who presided, and by others prominently identified with the mining and metallurgical industries, as well as by representatives of the governments of various parts of the Empire.

GEO. C. MACKENZIE, General Secretary.

R. O. WHEATLEY,

Associate Secretary.

923 Drummond Building, Montreal, April, 1928.

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MINING COAL UNDER THE SEA IN NOVA SCOTIA WITH NOTES ON COMPARABLE UNDERSEA COAL-MINING OPERATIONS ELSEWHERE*

By Francis W. Gray**

(Member, C. Inst. M. & M.: President, Min. Soc. N.S.)

(Sydney, N.S., Meeting, September 9th, 1927)

Introduction

Extensive undersea coal-mining has developed on both the east and west coasts of Canada, and at this time some four million tons, or 25 per cent of the total output of Canadian coal, representing a pit-mouth value of \$14,000,000 per annum, is coming from coal seams lying under the sea. Of the coal now being mined in Nova Scotia, 55 per cent comes from under the sea, and this proportion is likely to increase.

The Province of Nova Scotia is at this date receiving in royalties paid upon coal removed from undersea areas a sum exceeding \$400,000 per annum.

The subject is, therefore, one of special interest to the Mining Society of Nova Scotia and the Canadian Institute of Mining & Metallurgy, and while this paper will deal chiefly with undersea coal-mining in Nova Scotia, an attempt has been made to summarize the development of mining practice in the winning of undersea coal in other parts of the world, with the hope that such a recital may assist in the assessment of local problems.

The connection between the mining of coal under the sea and an insular or maritime geography is obvious in regard to Great Britain, Japan, Vancouver Island, Australia, and Cape Breton Island, and also in regard to the curious conformation of Chile; the tendency towards growth of a mercantile marine and naval eminence is equally obvious where coal is mined at tidewater. But, there is the further and less favourable significance that the mining of coal under the sea is associated with poverty rather than with abundance of coal reserves.

^{*}This paper was presented at the Annual Meeting of the Mining Society of Nova Scotia, Baddeck, C.B., June 21st-22nd, 1927.

**British Empire Steel Corporation.

From this it follows that undersea coal reserves are of exceptional economic value to the countries of their occurrence, and their maximum utilization is a subject of first-rate importance in a situation so unusual as that of the Cape Breton submarine coalfield, which, besides being the only coal deposit on the Atlantic coast of the American continent, is the only really large deposit of bituminous coking-coal in Canada east of Alberta.

There are not many instances of as great a stretch of populated territory so poorly provided with coal deposits, and this fact attaches to the submarine coal-areas of Cape Breton unique importance from a national and cultural standpoint.

A summary of the extent of coal production from under the sea in various quarters of the world is, in round figures, as follows:

Countries	Long tons per annum		
Great Britain: East Coast West Coast Scotland Canada: Nova Scotia Vancouver Island Australia Total British Empire Chile Japan		6,600,000 3,900,000 200,000 10,700,000 1,200,000 1,500,000 13,400,000	

The literature on undersea coal mining is not large, and is almost entirely to be found in the *Transactions* of the Institution of Mining Engineers, and in the proceedings of our own Society. A bibliography of the main references is appended.

UNDERSEA COAL MINING IN NOVA SCOTIA

Coal is mined from under the sea in three separated localities in Nova Scotia, namely, beneath the Atlantic at the entrance of and in the vicinity of Sydney harbour on the eastern shores of Cape Breton island; beneath the gulf of St. Lawrence off Inverness County on the western coast of this island; and beneath the upper reaches of the bay of Fundy in the Joggins Mines district in Cumberland county.

The extent of the undersea coal is unknown. Carboniferous rocks are present in the Maritime Provinces and Newfoundland over an area roughly describable as a parallelogram 300 miles by 200 miles in extent, extending from Vanceboro on the Maine border to the mouth of Baie Chaleur, New Brunswick, thence to the head of St. George's bay, Newfoundland, proceeding southwards to Isle Madam, and thence westwards to Vanceboro (See map of Nova Scotia, Figure 1).

This parallelogram takes in the lower gulf of St. Lawrence (including Prince Edward Island and the Magdalens) the Cabot strait and the Atlantic shore of Cape Breton Island, an area corresponding, very approximately, with the lowlands where the coal-seams were laid down.

Dr. W. A. Bell, of the Geological Survey of Canada, in contradistinction to earlier geological opinions, believes that the coal deposits in Nova Scotia were laid down in river valleys and flood-plain lakes (not necessarily connected) and that the sediments in the various coal basins were not synchronously laid down. (1)

The Minto coal horizon in New Brunswick is correlated as to geological age by Dr. Bell with the lowest seam in the Sydney coalfield, but there is no evidence that the Productive Coal Measures (such as overlie the lower or 'Millstone Grit' measures in the Sydney field) were ever laid down in the New Brunswick area. Therefore the possibility of the presence of workable coal seams under the Permian rocks of Prince Edward Island is not promising. (2)

Recent deep borings in Prince Edward Island, in search for oil, indicate that the Productive Coal-Measures are absent below the Permian.

⁽¹⁾ The Southern part of the Sydney Coalfield, by A. O. Hayes and W. A. Bell; Geol. Survey of Canada, Memoir 133, pp. 5 and 6.
(2) Correlation of the Minto Coal Horizon; Geol. Survey of Canada, Summary Report, 1923, Part CII.

See also "Coalfields and Coal Industry of Eastern Canada;" Mines Branch, Dept. of Mines, Ottawa, Bull. No. 14, page 31.

The Cape Breton Island coalfields are predominantly submarine, only the fringes of the several basins remaining above sea-level. In several instances the apexes of the basins outcrop at the shore-line, with insufficient strata-cover to permit entrance by mining in some of the coal seams. Seams probably exist with outcroppings entirely under the sea.

THE SYDNEY COALFIELD, CAPE BRETON ISLAND

The Sydney coalfield, originally one continuous deposit, is divided by folds of the strata, running parallel to each other, into three main synclinal basins, with a general dip of the measures seawards, except as modified along the folds (See Figure 2).

The folds are not severe, and in one instance (the Glace Bay—Sydney Harbour basins) workings have proceeded under the sea from one basin into that adjoining, surmounting the minor anticline. There is reason to surmise that the folds diminish in size as they proceed seawards. This condition indicates that in future submarine workings, further seawards than those of present date, the probable smaller extent of the folds, combined with increased thickness of strata cover, will enable the mine projections to ignore the folds as obstacles between the several basins, at least in regard to the central part of the field, where the continuity of the seams over a sea-frontage of some 30 miles is fairly well established by cliff exposures and actual mining in land and sea areas.

Along the greater part of this sea-frontage, the workings have entered undersea territory upon exhaustion of mineable coal in the land area, and with two exceptions, which will be later referred to, none of the collieries have been developed primarily as submarine winnings.

A unique feature of the Sydney coalfield is the regular uninterrupted seaward-inclination of the measures and the included coal-seams, and the general absence of disturbances of the measures, except the changes of contour levels by the anticlinal folds and the shallow synclines lying in between.

No marked change in dip indicating the possibility of a flattening-out or rising tendency has to date been observed at the faces of the undersea workings, which have extended at various points to distances varying from a mile up to a maximum of $2\frac{1}{3}$ miles seawards. The seams dip at an average rate of six to seven per cent along the axes of the synclines. The sea-floor has an average inclination of 2 per cent. The stratacover increases, therefore, at the rate of some 250 feet per mile of travel of the workings. The maximum seaward extension of the workings is at the face of the main haulage-road of Princess colliery at Sydney Mines, where, at 11,800 feet from shore, the strata cover below the sea-bottom is 1,500 feet in thickness, this being to date the maximum depth below the sea attained in the Sydney coalfield.

There is, therefore, as the workings proceed seawards, rorelief obtainable from the increasing handicap of depth, or the consequent difficulties of extraction and underground haulage of the coal, difficulties more intractable than mere horizontal distance from the mine entrance on land.

Up to date, as previously noted, the submarine workings have been a more or less unplanned extension of land workings, proceeding, as was necessary, from known conditions to entirely unknown conditions. All the factors that are implied in the term 'workability' of the coal — quality, thickness, roof conditions, and so on — are unknowable in undersea mining until proved by actual workings, and this procession from land operations to submarine mining has been, in the nature of things, gradual, and admittedly tentative.

While the dominating hazard of undersea mining is the possibility of inundation from the sea, the really limiting factor is the cost of carrying ventilation currents, materials, and motive power, for long distances underground; and the complementary difficulty of transporting coal to the surface, all consequent upon the limitation of accessibility to points on land which become more remote as the working faces drive seawards.

The transport of men to and from the working faces within a statutory working-day, with sufficient allowance of productive labour period over and above the period occupied in walking and riding to the faces, is also a limiting factor.

It is evident, therefore, that the fullest use must be made of the entrances at the shore. The shafts must be large enough to admit the ultimate volume of air required for ventilation, and the main-airways underground must correspond in amplitude. As the recoverable tonnage seawards is necessarily available in some instances through one shore-locality only, and may represent as high as 200 years of potential production from one set of openings, the factors of amplitude and permanence in all equipment for ventilation, power-transmission, and underground transportation, assume an importance much greater than in land mining, calling for an entirely novel set of standards.

With the notable exception of the Princess pit at Sydney Mines, sunk sixty years ago, the No. 1B colliery of the Dominion Coal Company near Glace bay is the only shore-shaft deliberately designed primarily to win undersea coal, which is also of date sufficiently recent to include thoroughly up-to-date equipment. Even in this instance, the choice of site was restricted by the existence of land workings of much earlier date, and the new shaft connected with advanced underseaworkings of Dominion No. 1 colliery that had reached a point distant two miles from a shallow shaft sunk near the outcrop of the seam.

While, therefore, No. 1B colliery represents the commencement of a new stage in the development of undersea coal-mining in the Sydney field, it may be more correctly regarded as a transition type of colliery, foreshadowing the future colliery designed to win coal in what seems to us at this date impossibly remote submarine territory.

It is sometimes a corrective to our perspective of the future to look backwards.

In the Report of the Inspector of Mines of Nova Scotia for 1877, Dr. H. S. Poole essayed to forecast the future of undersea coal-mining in Nova Scotia, and the safeguards that legislation should impose, basing his views upon a careful review of submarine mining at that date in Great Britain. It is evident that Dr. Poole originated much of the legislation that has since regulated undersea mining in Nova Scotia, and that subsequent lease tangles occurred because his advice was not followed.

Enlightened and farseeing as this pioneer of undersea mining was, in asking for safeguards "to be interposed between selfish present interest and the future welfare of the coal industry", he remarks: "Not that it is likely in our day that the coal lying at extreme depths and in remote districts will be required, or that colliery establishments will be in a position to work outlying sections profitably, but for the benefit of those who will come after us, and who otherwise might justly reflect on our short-sighted policy and indifference to the country's future welfare."

Precisely fifty years have gone by, and two at least of the collieries with which Dr. Poole was acquainted, one of which he managed, are mining coal in 'outlying sections', and the operators have in one instance found the proceeding unprofitable.

The writer, in 1916 (¹), forecast that the limitation of coal-mining seawards in the Sydney field by cost of production would operate before any probable physical limitations or actual engineering difficulties of depth and remoteness from the land openings. This limitation by production costs is already well in sight in instances where the mine entrances, airways, and haulage passages, are conditioned by being the result of gradual seaward extension of land collieries. In several of these instances the continual seaward advance of workings has necessitated heavy expenditures in re-conditioning of main haulage-roads and in enlargement of airways.

Such betterments, while serving to prolong the period of profitable extraction through existing openings and thereby securing maximum use of present investments in mine development and equipment, will not permit economic winning of coal beyond a distance from shore that cannot at this time be accurately gauged, because so much will depend on thickness and quality of the coal-seams, and the depth of overburden. Mining under somewhat similar conditions, as at Whitehaven, has proceeded up to 4 miles from shore (see Figure 16), and in the upper seams of the Sydney field it may similarly be extended from existing openings. For the winning of coal beyond this as yet undemonstrated limit, new methods of access will doubtless be evolved.

⁽¹⁾ Coalfields and Coal Industry of Canada; Mines Branch, Dept. of Mines, Ottawa, Bull. No. 14, page 32.

As the new ideas of design incorporated in the construction of No. 1B colliery have resulted in lowered production cost, it is to be anticipated with confidence that new methods of access to outlying submarine areas will similarly reduce production cost, and indefinitely postpone the approach of a limitation of reserves by mining costs. It is necessary to make clear that, in the writer's opinion, such cost limitation as had to date been approached, is largely associated with inadequacy of existing shore openings.

Two other features of the mining practice to date account largely for the approach of a production-cost limit, namely, the rapid advance of undersea workings in areas where some 55 per cent of the coal has been left standing in support pillars; and, in general, the small use hitherto made of deep shafts and cross-measure tunnelling to minimize long underground uphill-hauls.

Both of these features are a result of experimental and largely unplanned progression of mining — extending over half a century — from land areas into undersea territory.

The resumé of the development of local undersea mining, which follows, is confined to a little over twenty years of personal observation, whereas the period under review is over fifty years. The details of the evolution of local practice are given in as condensed a form as is consistent with accuracy of record, and it is hoped that such conclusions and comment as are added to the record will be found to give full credit to the vision and workmanship of our predecessors; for, as Dr. Poole wrote fifty years ago, "Mining must to a large extent ever remain a matter of accumulated experience".

Whether our generation will, in the eyes of our successors, leave behind as good a record of workmanship as those who preceded us, it is not for us to say, but our problems are without cavil more onerous, and call for a wider range of technical attainment.

A summarization of the conditions touched upon in the following pages indicates that undersea mining in the Sydney field calls for observation, study, and research in several specialized branches of mining science, namely:

(a).—The local conditions of coal deposition, which requires, as a preliminary, study of the pre-Carboniferous floor of deposition, a subject that in turn calls for research into the conditions that delimited the basins wherein the coal-bearing rocks were laid down, and calls for a specialist in tectonic geology applied to the coalfield of Eastern Canada whose services have not yet been developed(1).

Questions of quality and economic availability of coal, allied to geological structure of the coalfields, are a part of this general problem.

(b).—The development of technicians of high-grade ability, for co-operation with practical coal-miners, in the unique problems of subsidence, of transportation, ventilation, and transmission of motive-power, that will accompany the development of what may eventually prove to be the classic instance of undersea coal-mining, both in regard to depth of strata-cover and seaward extension of mine-workings.

If the achievement of this generation is as great as its opportunity, in regard to the Sydney coalfield, it will indeed be noteworthy.

The Development of Undersea Mining in the Sydney Field

Mining under the sea was first attempted in the Sydney coalfield in 1867, when a slope was driven under the waters of Sydney harbour in the Victoria (Harbour) seam.

The site of this opening — the old Victoria mine — was not the most suitable, as it was near the maximum inclination of the measures in this district.

⁽¹⁾ With regard to historical geology and the sequence of tectonic movements deciding the presence of known and concealed coalfields in the Maritime Provinces, it is but fair to explain that the writer's reference to the necessity for a specialist in this regard "whose services have not yet been developed" was made without knowledge of Dr. W. A. Bell's recent studies, and, more especially, his communication to the *Transactions* of the Royal Society of Canada on "Outline of Carboniferous Stratigraphy and Geologic History of the Maritime Provinces". This paper discloses that in Dr. Bell the Geological Survey has now a specialist whose studies have been directed to elucidation of those broad outlines of geologic events which must precede intensive study of local geology and correlation. It is to be desired that the Geological Survey will be able to arrange that Dr. Bell shall continue to devote his energies to the coal geology of the Maritime Provinces.

At this date the south shore of Sydney harbour was described by Richard Brown as "a terra incognita from a geological point of view." The measures at the Victoria mine dip to the north at an angle of 25 degrees; those on the opposite shore dip $4\frac{1}{2}$ degrees east — the dips being almost at right angles to each other in shore-exposures separated by only two miles of water.

The Victoria mine was worked until 1897, when it was closed by the Dominion Coal Company, into whose possession the property had entered in 1893. Meantime the workings of the Princess colliery on the 'Main seam' had proceeded under the harbour to the lease boundary, one mile from shore, proving a continuance of the $4\frac{1}{2}$ degrees east dip at a distance of less than one mile from the steeply dipping Victoria workings. Such a condition suggested the presence of a fault to many observers. Richard Brown thought it evident "there is one. perhaps two large faults under the waters of the harbour", but, as his writings clearly indicate, Mr. Brown supposed that the Sydney Mines seams were "the very lowest in the coalfield", and did not apprehend the condition of progressive overlap, shown in Figure 11, that later researches have only recently disclosed. Mr. Brown was puzzled by the absence from the section at Low point (1) of the lower seams and sediments of the Morien Bay section.

In this instance, the study of the strata measurements and lithological considerations prevented this pioneer observer from accepting his own first impressions, for Mr. Brown remarks:

"Looking at the map, it may naturally be supposed that, if the measures on the north shore bend round to the east in their course under the harbour, they will meet and coincide with those running in an opposite direction from the Low Point shore; and, thus united, will form the end or segment of a large basin" (2)

which is, indeed, a precise description of the actual conditions.

Mention is made of these earlier theories because they have greatly influenced the development policy of the mining companies, particularly in regard to the direction of the Princess colliery enterprise, and because they serve as an indication of the small reliance that can be placed upon stratigraphical and lithological factors as a guide to correlation in a flood-plain deposit. (3)

⁽¹⁾ and (2) Coalfields of Cape Breton, page 35.
(3) "The Southern part of the Sydney Coalfield," Hayes and Bell, page 56.

The workings of the Victoria mine were cleared of water in 1919, and the deeps were driven from where they had stopped advancing at the shore-line, a further distance of 1,850 feet seawards; the profile of the slope being for the first 2,000 feet an angle of 25 degrees, for the next 900 feet, 10 degrees, and for the remaining 500 feet to the face, $7\frac{1}{4}$ degrees.

The development of a modern colliery had been planned at the Victoria mines site, and part of the equipment was actually installed, but war conditions deferred further work. In the meantime, the unified management of the coal properties which followed the formation of the British Empire Steel Corporation in 1920 led to the abandonment of this site in favour of a site in the vicinity of Low point (elsewhere referred to), where the strata is not so steeply inclined and a better grip of the submarine coal can be obtained.

During 1926 a reallotment of leased submarine areas was agreed upon, extending the territory of Princess colliery to the south shore of the harbour, or up to the boundary barrier of the Victoria mine.

The next in date, but more important, winning of undersea coal, followed the sinking of the Princess pit by the General Mining Association to the 'Main seam' at Sydney Mines. The sinking was commenced in 1868, and was completed in 1876, winning the seam at a depth of 687 feet. A detailed description of the Princess colliery is given when dealing with the Harbour seam elsewhere in this paper.

The workings of Princess colliery entered the submarine area in 1877, so that 1927 marks a half century of continuous undersea mining in this colliery; and the fiftieth anniversary of large-scale submarine mining in Nova Scotia.

It is further noteworthy that 1927 is the centenary of the commencement of extensive coal-mining in Nova Scotia, as it was on January 1st, 1827, that the General Mining Association acquired the Sydney Mines property.

While the situation of the Princess shafts, close to the shore, is evidence that the 'New winning' (as it is still locally named) was designed to serve an entirely submarine area, the General Mining Association seems not to have contemplated the likelihood of successful mining beyond a distance much exceeding one mile from shore; because, notwithstanding that

this pioneer Association had unique opportunities to acquire additional leases, it did not over a period of 75 years add to its original undersea leases, having a maximum extent of 1½ miles from shore.

As to this, Mr. Richard Brown stated: (1)

"For many years the Company worked only the whole coal, leaving all the pillars standing. The Company hesitated to begin the removal of pillars in this submarine lease, proposing to postpone such operations until after the whole coal should have been wrought as far as the limit to the extreme dip.

A further contemporary note states: (2)

"No pillars will be worked until the sea-boundary is reached, but afterwards it is intended to remove the whole of the seam, leaving the goaf to the dip.'

It is evident that no provision to preserve access through the goaf to the areas beyond the lease boundary was contemplated. The existence today of working faces a mile beyond the boundary referred to is sufficient commentary on this phase of the matter.

The next submarine winning was from the Hub seam at Table head in 1897, followed by working of the Gowrie seam under Morien bay, and a general advance into the undersea extensions of the Harbour and Phalen seams in the Glace bay area between 1903 and 1907. In the Waterford area, submarine mining commenced in 1910.

A tabular arrangement of dates of entrance into the undersea extension of the coalfield is given for reference purposes (see Appendix F).

Correlation and Description of the Coal-Seams of the Sydney Field.

The intersection of the visible portion of the Sydney coalfield by arms of the sea, and the division of mining operations arising from independent corporate activities, have made it locally customary to think of the field as divided into separate basins, but, for the purposes of this paper, the submarine tract will be treated as one continuous area of coal-bearing strata, about which little can be learned except from the

⁽¹⁾ Discussion of paper by A. A. Atkinson; Trans. Inst. Min. Eng., Vol. XXIII, 1902, p. 255, also pages 636 and 647.
(2) "Notes on the working of coal-mines under the sea, and also under the Permian Feeder of water in the County of Durham", by T. Bell; Trans. Manchester Geol. Soc., 1899.

progress of mine workings therein; and in which new coalwinnings will be most successfully planned by recognition of the submarine area as one that has not yet been prospected, and that must be judged in matters of engineering without too much reliance on the local tradition of mining as this has developed up to the shore-line.

Without attempting to discuss fully the intricate details of correlation of the seams in the coal-field, it may be stated that, while the seams are persistent, their thickness, and that of the included shale-bands, varies, and there is a tendency for shale bands to thicken, in some instances splitting the seams into two or more widely-separated portions.

Recently, palæontological evidence has been used by the Geological Survey in correlation of the seams, (¹) and this, added to the progress of the undersea workings and recent bore-holes put down in the Glace Bay area by the Dominion Coal Company, has enabled a more exact idea to be formed of the identity and continuity of the undersea seams than was until recently the case.

Dr. W. A. Bell, whose recent work on the Sydney coalfield has an important economic bearing, states that the coal-bearing strata of the Sydney field comprise roughly 6,200 feet of sediments, predominantly grey, and wholly of fresh-water origin. The known workable coals are restricted to the upper 1,900 feet of the present land representation of the series, and all lie within a zone of occurrence of *anthracomya* shells.

This 6,200 feet of strata, which Dr. Bell names the 'Morien series', and to which he assigns a position at the base of the Pennsylvanian, consists "of a laterally-changing alternation of sandstones and shales. The mass as a whole is built up of a succession of interlocking lenses, and not of a succession of uniform sheets." For this reason, writes Dr. Bell, "no reliable correlation of the coal seams can be based on a comparison of thicknesses, and lithology of individual beds", a conclusion that will not be disputed by those who have attempted a correlation.

^{(1) &}quot;The Southern Part of the Sydney Coalfield", by A. O. Hayes and W. A. Bell; Geol. Survey of Canada, Memoir 133.

In the central portion of the coalfield, namely that extending from cape Percy on the east to the narrow channel of the Little Bras d'Or on the west, and including the mining centres of Glace Bay, Waterford and Sydney Mines — i.e., all the presently-operated coalfield — the correlation of the seams is commencing to be generally understood. Little, however, is known of an exact nature, or from actual mining, of the seams and associated strata west of the Little Bras d'Or or southeast of cape Morien.

The largest concentration of undersea mining has developed in the central part of the Sydney field, near the town of Glace Bay. The promontory of Table head, through which there passes the axis of the flat syncline of the Glace Bay area, contains the semi-circular out-cropping of the highest seam exposed on land in this area — the Hub seam — this outcrop measuring between the cliff exposures roughly one mile, with an inland radius slightly exceeding a half-mile(1). The lower seams outcrop successively in concentric form, the outcrop of the lowest known workable seam having a maximum inland radius of five miles from Table head (see map of Sydney coalfield, Figure 2).

A double-compartment shaft (mines Nos. 2 and 9 of the Dominion Coal Company) was sunk in 1900 half a mile inland from Table head, upon the pavement of the Hub seam. One side of this shaft reaches the Harbour seam at 402 feet deep; the other compartment continuing to the Phalen seam, reached at a depth of 842 feet.

The following section, arranged from the strata record in the No. 2 shaft, and from a borehole sunk from the pavement of the Phalen seam at No. 1B shaft, slightly over a mile to the north-westward, may be taken as typical of the submarine tract. The section is especially interesting as showing the spacing of the seams in the only part of the Sydney field where three seams have been simultaneously mined in superimposed planes under the sea. These seams are indicated by capitals.

⁽¹⁾ The name 'Hub' is said to have been given to the colliery at Table head because it was placed in the centre of the concentric outcroppings, but the writer has not been able to confirm this rather natural assumption.

Section of Undersea Coal-measures in the Glace Bay district of the Sydney Coal-Field, Nova Scotia

Zone	Seam	Coal and Measures (feet)	Total depth (feet)	
Anthracomya {	HUB HARBOUR Boutilier Back Pit PHALEN.	9 ft. 0 in. 380 ft. 0 in. 5 ft. 8 in. 248 ft. 0 in. 3 ft. 9 in. 79 ft. 0 in. 3 ft. 0 in. 112 ft. 0 in. 7 ft. 0 in.	394 ft. 8 in. 646 ft. 5 in. 728 ft. 5 in. 847 ft. 5 in.	
Linopteris obliqua Neuropteris gigantea	Gardiner	131 ft. 0 in. 3 ft. 0 in. 427 ft. 0 in. 0 ft. 6 in. 180 ft. 0 in. 3 ft. 0 in. 3 ft. 0 in. 3 ft. 0 in. 1,900 ft. 0 in.	981 ft. 5 in. 1,408 ft. 11 in. 1,630 ft. 5 in.	

It will, following this very condensed summary of the present status of correlation of the seams, be convenient to describe the actual mining so far carried out under the nameheadings of the seams in the No. 2 shaft section.

Hub Seam:

This seam, the highest preserved in the Glace Bay area, was mined at the shore outcroppings to provide coal for the Louisburg garrison during the English occupation of 1745-49. A mine fire in 1872 caused extensive slagging of the shales at the shore-line.

The small land-area of the seam has been entirely worked over, at first from shore-levels, and later by a shaft, 120 feet deep, situated close to the shore-cliff. The seam varied in thickness from eight feet to nine feet under the entire landarea, and seawards for a distance of 4,000 feet. The main

deeps are driven 8,200 feet on the full dip, which averages 3 deg. 23 min. for a distance of 4,600 feet from the shore. At this point the seam is level for 80 feet, thence rises at an angle of 5 degrees for 130 feet, followed by a falling angle of 1 deg. 43 min. for 500 feet. From this point to within 200 feet of the face the grade averages 4 degrees, but the last 200 feet to the face of the deeps dips 6 degrees, which is normal for this area.

The change to a rising grade coincided with the encountering of a reverse fault with a throw of five feet six inches, and with a sudden thickening of a parting in the centre of the seam. This parting, barely perceptible in the land area, steadily thickened until at 4,400 feet from shore it measured twelve inches. At 4,500 feet, the parting having increased to three feet, mining was continued on the upper half of the seam, 4 ft. 6 in. of excellent coal. The position of the lower leaf of the seam was proved by borings, disclosing apparently a lens-shaped inclusion of shale and coal, 25 feet in depth at the thickest part of the lens, and of unknown lateral extension. In the next 700 feet this decreased to 8 ft. 6 in. of shale and coal with the lower leaf of coal 3 ft. 4 in. in thickness, and was maintaining this section when the mine was closed.

The splitting of the seam, with which was associated the alteration in its inclination, increased the cost of extraction, and caused difficulties with drainage and haulage, already sufficiently onerous by reason of the long uphill-haul to the shaft bottom.

The resulting high cost of production caused stoppage of production in 1918 and eventual closing of the mine, which has been allowed to fill with water. The undersea workings, 475 acres in extent (of which 245 acres was thick coal, and the remainder only the upper 4 ft. 6 in. of the split seam) are 'pillar and room.' No pillars have been drawn. Depth of solid strata-cover at the face of the deeps is 560 feet, covered by 68 feet of water at mean sea-level.

With the exception of the semi-circular area inside the land crop, previously mentioned, the whole of the seam lies under the sea, in a narrow tongue-like syncline, estimated to measure $2\frac{1}{2}$ miles between the undersea outcroppings and $1\frac{1}{2}$ miles between the 200-ft. cover contour-lines at the point where mining was suspended.

The unmined undersea-area will in all probability be some day reached by a rising cross-measure drift from workings in the lower seams, one of which, the Phalen seam, lying 831 feet below (measuring from pavement of Hub to roof of Phalen) has been mined seawards to a point 400 feet in advance of the abandoned faces in the Hub seam.

The equivalent of the Hub seam on the Waterford side of the Bridgeport anticline, known as the 'Barrasois', was opened from the outcrop by the General Mining Association about 1870 near the present No. 14 colliery of the Dominion Coal Company and driven 1,130 feet, just entering the submarine area, but development was not further proceeded with. The section in this slope, in descending order, is 2 feet 8 inches of coal, 2 feet of clay, and 6 feet of coal. Two miles west of these slopes, the outcrop section shows 1 ft. 5 in. of inferior coal, two inches of splint, and 5 ft. 7 in. of good coal.

A cross-measure drift from the workings of the Harbour seams proved the Barrasois seam at a point 5,000 ft. seawards from shore, the section being here 1 ft. $4\frac{1}{2}$ in. coal and splint, band of splint $4\frac{1}{2}$ inches, coal 4 feet. (1)

The interval between the Harbour and the Barrasois seams is 384 feet at the drift intersection, or four feet more than the interval between the seams at Table head. This close agreement is surprising in view of the tendency of the measures between the coal-seams to thicken in a westerly direction. Strata intervals are not a safe guide to correlation in the Sydney field.

There is present in the Waterford area, below the Barrasois, a thin seam having a section, in descending order, coal 8 inches, clay and shale 1 ft. 7 in., coal 2 ft. 4 in., lying 40 feet below the Barrasois seam, which may represent the lower leaf of the Hub seam, the splitting of which has been noted as occurring off Table head.

⁽¹⁾ Note: In driving this drift through a sandstone layer, the drillers holed into a cavity containing water under heavy pressure, so heavy that the drill was forced backwards, and the drillers were compelled to escape from what at first appeared to be a serious inbreak of water. Later it was discovered that the cavity was in the nature of a 'geode' lined with fully-developed calcite crystals, with characteristic cleavage. The crystals were a dirty grey in colour, having evidently been deposited from water stained by passage through coal and coaly-shales. Unfortunately the dimensions of the cavity are not known.

The evidence suggests the continuity of a four-foot seam equivalent with the Hub over the whole area from Table head to No. 14 colliery, a distance of six miles. A considerable part of this area will be unmineable because of lessened cover over the course of the Bridgeport anticline.

West of the Waterford collieries, and approaching Sydney harbour, the Hub seam is believed to split up into an assemblage of thin coal seams and shales represented on Cranberry head by the Chapel Point seams. Further west these seams apparently come together as the Stubbert seam, which has a land outcropping across the end of Boularderie island of two miles, and is from 8 feet to 9 feet in thickness, with two included clay-partings. A large reserve of submarine coal is present in the Stubbert seam, if its seaward continuation maintains the thickness shown at the outcrop, but nothing is as yet known as to this. The dips are slight, probably under 4 degrees, a condition that would permit of workings proceeding a very long distance seawards without gaining undue depth of cover. A very large area of the Stubbert seam may be available under the opening of the Great Bras d'Or in the direction of the Bird islands. Much will depend upon the extent of the folding in the Great Bras d'Or syncline, which does not, from available indications, appear to be severe.

Harbour Seam:

Named from its outcropping on the shore of Glace Bay harbour, this seam has been worked in the undersea area more extensively than any other in the Sydney coalfield. Following exhaustion of the land area in the Glace Bay district, it was followed seawards by workings tributary to No. 2 shaft, entering the submarine with 370 feet of cover, and is developed most extensively to the southeast on the level-contour of the seam. In this direction the seam is worked to the 200 ft. submarine cover-line. The maximum undersea cover so far worked under is 663 feet of strata with water cover of 43 feet, reached at a distance of 4,000 feet on the full dip of the seam.

A submarine area of 1,035 acres, averaging 5 ft. 6 in. in thickness, has been worked pillar-and-room, no pillars having been drawn.

The measures overlying the Harbour seam for ten feet above are very weak, and in 1921, because of this, and consequent upon inability to overtake upkeep-work during the war period, extensive repairs to airways and main haulages were required. The workings in the Phalen seam below having advanced into the submarine so far as to have attained a depth of 1,000 feet — the cover at which it is considered pillars can be safely drawn — it was decided to suspend the mining of coal in the Harbour seam to permit extraction of Phalen pillars directly below, as elsewhere explained, and no coal has been mined since 1921.

East of the workings tributary to the No. 2 shaft the seam has not been mined. The outcropping is submarine in its entire eastward course and the seam is accessible only by cross-measure drifting. It is proved by a drift rising from the Phalen seam workings of No. 6 colliery, the thickness and quality being unchanged where encountered by the drift at a point distant 4,700 feet from shore, and 10,100 feet to the eastward of No. 2 shaft workings, with 447 feet of strata cover.

In the Morien basin, the Harbour is believed to be represented by the Blockhouse seam, 9 feet in thickness. This seam, having an inland extension of one mile, and a width between shore-outcroppings of half a mile, has been worked out under the land area, but has not been followed to sea because of insufficient strata-cover. It may eventually be worked by a rising drift from the workings of the Gowrie seam below (at present full of mine water), but the narrowness of the syncline will require an extension seawards disproportionate to the width of the workable area.

Between the No. 2 shaft area and the area being now mined in the underlying Phalen seam in the territory allotted to No. 1B colliery, the Bridgeport anticline will lessen the undersea cover of the Harbour seam. It will in all probability be reached by a cross-measure drift from the No. 1B Phalen workings, later described, and the unusual features of the haulage roads in the Phalen seam will doubtless be repeated in the overlying Harbour seam, but much further seawards.

A large area of the Harbour seam will be ungettable because of the undersea outcropping of the seam and shallow submarinecover between the Glace Bay and Waterford areas.

On the westward side of the Bridgeport anticline, in the Waterford district, the seam is worked under the name of 'Victoria', in collieries Nos. 12 and 14 of the Dominion Coal Company. These mines, one mile apart, were opened in the period 1907-9 and entered the submarine area very quickly, as the land area is small in extent. The site of these openings is approximately midway between the Bridgeport anticline and the axis of the sharp syncline that runs close to the southern shore of the entrance to Sydney harbour.

The seam is entered by slopes from the outcropping. The No. 14 colliery deeps, which entered the submarine in 1910, are advanced 6,000 feet from the shore-line seawards. The inclination of the seam is 17 degrees for 1,400 feet from the outcrop to the shore-line, lessening seawards, the last 1,000 feet of the deep dipping $7\frac{3}{4}$ degrees. The average inclination is $9\frac{1}{2}$ degrees. The seam is 6 ft. 6 in. in thickness. At the face of the deeps there is 994 feet of solid cover and a depth of water of 57 feet overlying.

The colliery immediately adjoining to the westward — No. 12 colliery — entered the submarine in 1911. The deeps have an average inclination of 10 degrees. From the outcrop to the shore-line, a distance of 2,900 ft., the seam dips 11 degrees. The deeps extend 4,500 feet seawards from shore, the last 1,000 feet dipping $9\frac{1}{2}$ degrees. Depth of cover at the face of the deeps is 1,196 ft. of strata, and 56 feet of water. The seam averages 6 ft. 4 in. in thickness in this colliery.

These two collieries are working the Harbour seam under the sea over a combined frontage of 8,400 feet.

To the westward of No. 12 colliery, occupying the territory intervening between the boundary of this mine and the area underlying the entire extent of the Sydney harbour entrance from shore to shore (now allocated to the Princess colliery on the northwest shore) is a block of undersea coal, with a minimum seaward frontage of one mile, that will probably be won at a future date by a shaft near the shore at Low point. This projection of the coast-line includes the fragmentary outcroppings of the equivalents of the two seams above the Hub

seam, known as the *Aconi* and *Bonar* seams in the Sydney Mines-Bras d'Or area, so that a shaft sunk at this point would intersect all the known seams.

The Low point site resembles in its command of the coal-seams of the Sydney Harbour syncline the site at Table head in the Glace Bay area, and because of the steeper inclination of the coal-seams — 15 degrees on the eastern side and 24 degrees on the western side of the headland — offers probably the most suitable site in the whole coalfield for employment of a deep shaft and cross-measure drifting.

The working of the Harbour seam at Victoria colliery has been mentioned when reciting the development of undersea mining in the Sydney field.

Sydney Mines Area:

Princess and Florence Collieries.—The Harbour seam has been more extensively worked on the Sydney Mines side of the Harbour than elsewhere in the field.

The Princess and Florence collieries are mining an area separated by an inter-colliery barrier. The seam varies from 5 ft. 8 in. thick at the eastern extremity of the Princess workings to 4 ft. 8 in. at the face of the Florence deep, a developed frontage of $3\frac{1}{2}$ miles, showing a tendency to maintain its thickness at sea, but thinning in a southwesterly direction towards the land outcropping.

The general dip of the seam over the whole area named is 4° 50′ northeast, but this has decreased to 3° 17′ in the last 4,000 feet of the Princess deep.

The Princess shaft is situated 9,800 feet from the land outcropping of the seam, and 800 feet from the shore-line, reaching the seam at 687 feet dep. The face of the main deep is 11,800 feet from shore, with a cover at this point of 1,456 feet of solid strata. The distance to the Waterford shore from the nearest point in the Princess workings is 8,000 feet.

The Princess shaft-sinking was commenced in 1868, and completed in 1876. A feeder of salt water was encountered at a depth of 270 feet from the surface (220 feet below sealevel) which necessitated lining of the winding and pumping

shafts with cast-iron tubbing from the surface downward for 300 feet. The hoisting shaft is 13 feet in diameter, and the air-shaft 11 feet. (1)

An undersea area of 2,960 acres has been worked over through these shafts.

Bord-and-pillar workings entered the submarine in 1877, with an overhead cover at the shore-line of 690 feet. Rooms were driven 17 feet wide and narrow-work places $6\frac{1}{2}$ to 7 feet wide. Pillars were left 40 feet wide by 123 feet long. In 1913 the pillars were increased to 66 ft. by 100 ft., with rooms driven 18 ft. wide and 14-ft. cross-cuts. Later the pillars were made 70 ft. by 100 ft., which is present practice where roomwork is adopted.

The recovery of pillars, at first in a tentative manner, commenced in 1899 at a depth of 800 feet and approximately 480 acres of pillar-coal has been removed.

In 1899, also, longwall mining was commenced, and seven acres were extracted, but for some reason the method was discontinued.

Longwall mining was recommenced in 1922 and has so far progressed that 85 per cent of the Princess output is, at the commencement of 1927, coming from longwall faces, some 400,000 tons having been so far got by this method. (2) It is expected shortly to use longwall entirely in this mine.

The Florence colliery is a slope opening (commenced in 1902), the mouth of the slope being 3,700 feet distant from the shore-line. The face of the slope is 8,500 feet from shore, with levels extending 4,700 feet in an easterly and 3,600 feet in a westerly direction. There is 876 feet of strata cover and 60 feet of water at the face of the slope, which is the point of maximum penetration seawards.

An area of 965 acres has been worked-over seawards by pillar-and-room method. At this time 15 per cent of the output is coming from longwall faces, and it is hoped to increase to 60 per cent. Longwall is not at present employed where

⁽¹⁾ Submarine Coal-mining, by Richard H. Brown; Trans. Mining Society of Nova Scotia, 1904.

⁽²⁾ J. J. McDougall, Mining Society of Nova Scotia, June, 1926.

the undersea cover is less than 800 feet, and in a portion of the Florence area the cover is less than this. No pillars have been as yet removed in the Florence mine.

The Princess area has a soft pavement which swells with exposure to moisture and 'heaves' under roof pressure to such an extent that maintenance of roads in the 'whole', and recovery of pillars in the 'broken', became virtually impossible in the advanced submarine areas, so that recourse is had to the longwall method.

West of the Florence workings, the Harbour seam diminishes in thickness in a general southwesterly direction, and on the shore of the Great Bras d'Or entrance is present as a series of coal-and-clay bands. A large area of the seam underlying the land is probably so deteriorated as to be unworkable, and little knowledge exists of the submarine extension off point Aconi; but immediately adjoining and west of the Florence workings there is a shore frontage exceeding three miles in length where the seam is lessened in thickness, but is probably of suitable thickness for longwall mining under the sea. As the deterioration and splitting-up of the seam in a southwesterly direction is associated with approach to the original limit of deposition, the reverse condition should be encountered seawards, especially in a general northeasterly direction.

There is a possibility that the basin of deposition may itself terminate before reaching the seaward prolongation of the cape Dauphin mountain fault, in which case a replacement of coal horizons by barren sediments occurs under the sea. There is, at the present stage of penetration of the undersea coal by workings, no evidence of such termination.

Phalen Seam:

This seam, averaging in the central part of the coalfield from 6 ft. to 7 ft. 6 in. of clean coal, and of consistently good quality, has naturally been extensively worked, particularly in the Glace Bay land area (approximately 6,300 acres), which has been entirely worked over by pillar-and-room method, a partial recovery only of pillars having taken place. Final

removal of pillars preparatory to abandonment of two old collieries is the only land-mining of this seam now taking place.

The seam was first attacked along the outcropping, and, later, shallow shafts were sunk to the deep; but, generally speaking, all the coal mined from this seam has been drawn from the deep, except a small area to the rise of No. 2 shaft.

The seam has been mined under the sea in the Glace Bay area over a shore frontage of seven miles, by four collieries: namely, commencing on the west and proceeding eastwards, collieries of the Dominion Coal Company, No. 1 (A and B) and Nos. 2, 4, and 6. Barriers of unworked coal 300 ft. thick are left between the individual mines in the submarine extension. In the neighbourhood of Glace bay, between the area allotted to No. 2 colliery and that allotted to No. 4 colliery, a block of unmined coal is being left in the submarine tract with an unmined land-extension a half-mile wide, and a projected length seawards of two miles (as shown on Plan No. 2) to provide access to the outlying seaward area in the event of inundation from the sea.

It will be convenient to consider the Phalen undersea workings in the western and eastern divisions created by this precautionary reserved block, the conditions differing in these divisions.

Figure 4 shows the development of the Phalen in the western division. The seam in No. 5 colliery (an entirely land area) and in collieries Nos. 1 and 2 (areas partly land and partly submarine) about 1914 became virtually exhausted under the land.

The outcropping of the seam in the area allotted to No. 1 colliery passes under the sea, and is deflected seawards into a sharp loop by the Bridgeport anticline. The seam is mined under the sea to the 180-foot cover-line, which, together with the level-contour-line curves in conformity with the anticlinal fold. The workings entered the submarine in 1905, the maindeep parallelling the shore. The face of the deep having reached a distance of 15,000 feet from No. 1 shaft, onerous conditions arose in regard to ventilation, haulage, and travel of workmen underground.

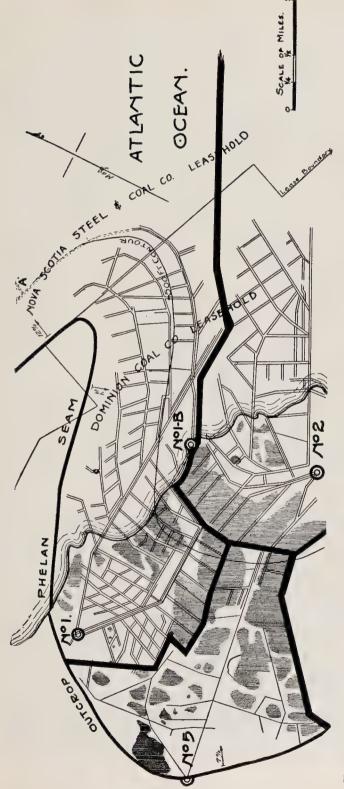


Figure 4.—Development of Phalen Seam in western portion of the Glace Bay area. Showing progress of workings in Dominion Coal Company's No. 1B Colliery over the Bridgeport Anticline.

The workings of No. 2 colliery, separated by a barrier of solid coal, had about this time developed so as to cover a very large area, and the need was felt for an exit additional to the separate coal-hoisting shaft and man-hoisting shaft at No. 2 colliery, that would serve in the event of fire or explosion.

The barrier pillar at the shore, at a point roughly midway between No. 1 and No. 2 shafts, afforded the only suitable site for a new shaft. A circular shaft 670 feet deep and 12 feet diameter was completed in 1921, to serve as a return-air shaft and man-hoisting shaft for No. 1 colliery, and as an emergency exist from No. 2 colliery. The passage-way through the barrier is 140 feet in length and six feet by eight feet in section. 'Explosion-doors' have been installed in this passage-way, which it is hoped will, in the event of an explosion in either of two mines so connected, resist the force of the blast, and preserve the escape-way intact. There are four doors, arranged in pairs, constructed of cast steel, set in cast-steel frames, secured into a heavy reinforced-concrete ring. The doors are designed to resist a pressure of 700 lb. per square inch (see Figure 5).

At the date referred to, 1921, the consolidation of the undersea lease-areas consequent upon the formation of the British Empire Steel Corporation was effected. The winning of the seaward coal being no longer hampered by lease limitations, it was decided to sink another shaft and develop a new colliery designed on a scale commensurate with the opportunity. A full description of this colliery, known temporarily as No. 1B (Figure 6a), has been contributed to the *Journal* of the Engineering Institute of Canada by Mr. Alex. Hay, Asst. Mining Engineer of the Dominion Coal Company, to which the reader is referred. (1) The salient features of the colliery, condensed from Mr. Hay's description, are as follows:

The shaft is 670 ft. deep, and 31 ft. 2 in. by 13 ft. 4 in. in section. There are three compartments, two used for coalhoisting and one as a pipeway. A collar of concrete extends from the surface for a depth of 32 feet. At 34 feet from the surface, the 14-inch shaft column discharges the mine water into a rock tunnel, 5 ft. by 8 ft., opening at a point in the face

⁽¹⁾ No. 1B Colliery of the Dominion Coal Company, by Alex. L. Hay; Bull. Can. Eng. Inst., Jan., 1926.



Figure 5.—Explosion doors between workings of No. 2 and No. 1B collieries of Dominion Coal Company. Designed to resist a pressure of 700 pounds per sq. inch.

of the shore-cliff 9 feet above mean sea-level. The water pumped from this shaft is all made in the rise workings, the undersea workings to the dip being virtually dry.

At 112 feet from the shaft-bottom, an opening into the Back Pit seam connecting with the Phalen seam serves as a permanent return-airway.

Owing to the close proximity of worked-out areas, the shaft bottom and approaches are strengthened by massive steel and concrete construction (see Figure 6b).

The main-haulage road, following as it does the level contour of the seam, swings through an arc of 137 degrees, a condition lending itself to the use of electric trolley-haulage. The haulage road, at this date, extends inbye three miles from the shaft bottom, and where it intersects the axis of the anticline it has a cover of 474 feet below the sea-floor. The road is driven 8 ft. by 20 ft. with a finished clearance of 7 feet by 18 feet. Sixty-pound rails are used, laid in a double-track of 36-inch gauge. The mine-cars are of 2-ton capacity, with solid ends for avoidance of dust. Sixteen-ton Goodman electric trolley-locomotives are used.

The projected line of the main haulage-way described extends under the sea for 4.3 miles from No. 1B shaft to the westward, where it will reach the line of the barrier separating No. 1B workings from the projected territory of a colliery later to be sunk on the eastern end of the Waterford area, where the Phalen seam is mined under the name of the 'Lingan' seam.

The roof, chiefly soft shale, is supported by steel rails, resting upon black-spruce props, embedded in sidewall packs, finished with a coat of 'gunite'. Later construction has employed solid-concrete sidewalls. This was done to prevent spalling of the coal pillars, to reduce lodgment of dust to the minimum, and to make a fireproof road to conform with the regulations governing the use of electricity in coal-mines. Stone-dusting is employed.

As this is the first employment of electric trolley-haulage on any scale in the Sydney coalfield, every precaution has been taken to minimize sparking and to lessen the dust danger. The whole course of this haulage is in intake air.



The second headland is the Waterford shore. The Bridgeport anticline underlies the bay beyond the first headland, on which the colliery is situated. Figure 6a.—Dominion Coal Company's No. 1B Colliery, situated near Glace Bay.

The ventilating fan is a 5-ft. by 10-ft. 'Sirocco', designed to deliver 300,000 cu. ft. of air at 6-in. w.g., a quantity of air not yet required.

The present air-shaft, sunk before the consolidation of the lease areas had made it possible to plan a winning of the importance that has since developed, will at a later date be replaced, as the seaward workings increase in area, by a new shaft of larger area, a necessity that may not arise for ten years to come.

The colliery was completed in 1924. It was designed to produce 2,500 tons of coal per 8-hour shift, but has much exceeded this, averaging at this date 2,900 tons per shift.

The coal believed to be available from workings contributory to this shaft should give it a producing life of from 125 to 200 years, depending on such limits to extraction as experience alone can disclose.

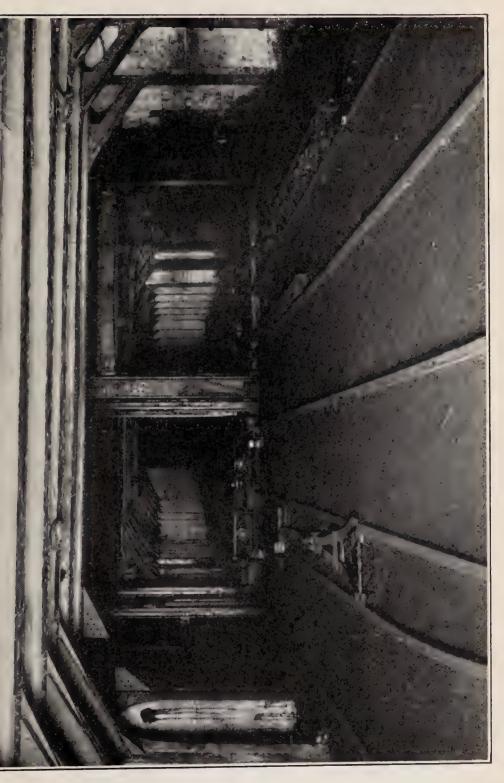
The Phalen seam only is being mined to the No. 1B shaft, but the Hub and Harbour seams overlying can be mined by rising drifts from the Phalen workings when these have advanced seawards sufficiently for this purpose. The Emery seam, lying 130 feet below, could also, in the submarine area, be mined to this shaft, but nothing is known as to its character in this vicinity.

No. 2 colliery of the Dominion Coal Company wins the Phalen seam in the adjoining area to the eastward. This territory occupies the flat syncline of the Glace Bay area, and along the axis of the trough there is a tendency to weakness of the immediate roof.

The submarine area was first entered in 1906, and to date 2,080 acres have been worked over by room-and-pillar method. Maximum seaward penetration is 8,400 feet, at which point a solid cover of 1,340 feet, and water-cover of 78 feet, exists. The seam is here 7 ft. 2 in. in height, and inclined at an angle of 6 degrees.

Pillar-drawing has been recently commenced in the submarine area in this colliery under an overhead cover of 1,000 feet, this being the only instance of pillar removal in the submarine area of the Phalen seam. As yet pillar-drawing has taken place only on a small scale, and more or less experimentally.





The pillars were made 45 ft. by 75 ft. If care is taken in timbering and drawing timber from the waste, about 95 per cent of the pillar-coal can be got. Some difficulties have been experienced in achieving complete removal of the timber from the waste, which throws the weight forward on the standing pillar, and there has been difficulty in getting the miners to work more than one pair in a pillar, resulting in a slow advance of the face. With a more extended waste, it is hoped that freer roof-action will follow. The operation would be facilitated by more rapid advance of the face.

The output of this colliery, about 4,000 tons per 24 hours on double-shift, comes entirely from under the sea.

On the eastward side of the separation barrier previously mentioned, the Phalen seam is mined by two collieries, No. 4 and No. 6 of the Dominion Coal Company, separated by the usual inter-colliery barrier.

No. 4 colliery (Caledonia mine) is the submarine extension of a colliery which was sunk in 1866, from which some fourteen million tons of coal have been mined to date.

The submarine area was entered in 1905 and 1,260 acres has been worked pillar-and-room. Maximum seaward penetration is 9,600 feet, with maximum solid cover of 1,147 feet and 72 feet of water. The seam is here 6 ft. 6 in., and is inclined seawards at 6 degrees. The daily output is 2,000 tons on double-shift.

No. 6 colliery, entering the submarine area in 1907, has mined over 1,040 acres of undersea coal and has prospected the eastern portion of the Phalen seam. Maximum seaward penetration is 6,800 feet, with solid cover of 1,074 feet and 56 feet of water. The seam is 6 ft. 6 in., and dips 7½ degrees.

The coal to the eastward is of much higher ash and sulphur content than in the Phalen seam elsewhere, and mining has been discontinued for this reason. The course of the barrier between the adjoining No. 4 colliery and this mine has been swung to the eastward to include territory formerly allocated to No. 6 colliery, and the further eastward development of the Phalen seam must depend upon elucidation of the geological structure of the eastern end of the coalfield, to which reference is herein made in dealing with the characteristics of the Cape Percy anticline.

Phalen Seam in the Morien Area.—The Gowrie seam in the Morien area is believed to be equivalent to the Phalen seam. The land area adjoining the shore is mined out. Workings were driven under the sea in 1899 and have been extended 9,600 feet seawards. The first mile of workings was mainly drivage. A frontage of slightly under a half-mile was developed when the mine was closed through financial difficulties in 1909. The seam averages 4 ft. 10 in. in thickness in the undersea area.

Phalen Seam in the Waterford Area.—Known in this comparatively recently-developed area under the name of the 'Lingan' seam(1), the Phalen has been proved by actual mining over a sea frontage of five miles.

A dirt band makes its appearance in the seam at about 2,000 feet from the point where the outcrop emerges on the Waterford shore from under the waters of Indian bay, and gradually thickens, splitting the seam. In this vicinity most of the land area, from the outcropping to the shore, was mined by the Lingan colliery, opened by the general Mining Association in 1855 and closed in 1884. Levels were driven under the sea, reaching a point 600 feet distant from shore, with a cover at that point of 210 feet of solid strata. It is interesting to note that the main haulage-level now being driven out of No. 1B colliery from the Glace Bay shore is following the same course as these old levels.

The development of a new colliery, with an entirely undersea tributary area, will shortly be undertaken in this vicinity. The choice of the site of the shafts is limited by the existence of old workings, which, being full of water, will be avoided in the planning of new workings. The thickness of the seam, between 8 feet and 10 feet, and its quality, are here exceptionally favourable, and the effect of the anticline will be to permit a maximum penetration of the Phalen seam seawards, because it will to a useful degree offset the gain of depth caused by inclination of the seams seawards.

To the westward, the Lingan seam has been mined by collieries Nos. 15 and 16 of the Dominion Coal Company, situated immediately to the rear of collieries Nos. 12 and 14, mining the Harbour seam above.

⁽¹⁾ Said to be a corruption of 'l'Indienne'.

In No. 15 colliery, the roof conditions were made difficult by the nearness of the upper leaf of the split Lingan seam to the lower leaf of the seam which is being mined, the roof falling upwards in working to the upper leaf of the seam 7 feet above. This colliery, opened in 1910, was closed in 1925, the area seaward being allotted for future mining to the adjacent No. 16 colliery, and to the projected new mine previously referred to, between which the original No. 15 area lies. No. 15 colliery-workings did not enter the submarine.

The workings of No. 16 colliery are at this date (1927) at the shore-line, the seam having been mined by longwall method in the lift approaching the shore.

In the seam above, pillars have been extracted in the submarine area from the 700-ft. cover-line seawards. Further seaward extension of workings in the Lingan seam below must therefore first proceed beneath an area of standing pillars and later beneath an area of extracted pillars, and the question of mode of working in the lower seam is now under consideration.

West of the No. 16 colliery area, the Lingan seam splits into two widely-separated seams, the parting which first appears near Northern head having attained a thickness of 138 feet on the south-east shore of Sydney harbour.

On the opposite shore, at Sydney Mines, no seam of economic value corresponding to the Phalen, or its Waterford district representatives, is definitely correlated.

Because of this uncertainty as to the seam itself, the steep inclination of the measures where they go under Sydney harbour, and the fact that in the vicinity of Low point the lower leaf of the split Phalen seam will be over 1,600 feet deep at the shore-line, the further undersea development of the Phalen seam west of No. 16 colliery area cannot be forecast.

Morien Area:

In this separated portion of the Sydney coalfield the seams have been preserved in a very narrow trough, the northern edge of which dips 45 degrees to the axis, from which the seams rise at an angle of 5 degrees to the southern outcropping.

The trough extends — so far as the upper series of seams is concerned — four miles inland, and, where it dips under the waters of Morien bay, it measures only $2\frac{1}{2}$ miles across.

While the series of seams found in the central portion of the coalfield is repeated in this narrow tongue-like area, the thickness and quality of the seams is varied.

Evidence of strata movements is present in the coal-seams in the vicinity of the axis, where the coal has a foliated appearance, and is shown also by clay intrusions into the Blockhouse seam which appear to have filled cracks in the surface of the seam(1).

The syncline dips under the sea at an angle of $2\frac{1}{2}$ degrees, and, as the sea-floor is only gently inclined, conditions for extended mining seawards are favourable, but the basin is very narrow.

The two seams mined to date, the Blockhouse in the land area only, and the Gowrie in both the land and submarine area, are correlated respectively with the Harbour and the Phalen seams, and are elsewhere dealt with under these headings.

None of the other seams are known to be of a thickness or quality to make them economically valuable, but their characteristics are entirely unknown in the undersea area, and little prospecting has been done in the land area.

There seems no reason, judging from the presence of coal seams on the narrow peninsula of cape Morien, to doubt the continuity of the seams seawards for a distance of three miles from shore at least. Beyond this, there is no evidence from which the geological structure can be deduced. Study of the structure at Flint island, elsewhere mentioned, may assist conjecture.

Between the outcrop of the upper series of seams at cape Morien and the point where the strike of the Tracey seam crosses the shore-line, a distance of four miles along the coast, there is presumably a seaward extension of this seam which should be mineable from a shaft at the shore sunk to obtain a suitable depth of cover. The contour of the seaward continuation of the outcropping being unknown, and the seam itself

^{(1) &#}x27;On a peculiarity in the Blockhouse Seam, Cow Bay, C.B.' by John Rutherford; Trans. N. S. Inst. of Science, 1868-9.

not being of high quality, undersea mining of this lowest seam in the field could not be profitably undertaken at this stage of development.

There is no basis for conjecture as to the shape of the coalfield after it disappears under the Atlantic in this vicinity.

Seams below the Phalen Seam,

not to date worked under the sea:

The *Emery seam*, underlying the Phalen seam at a depth of 130 to 150 feet, and extensively worked in the land area of the Glace Bay district, is believed to continue with a submarine outcrop around the Cape Percy anticline and to be the Spencer seam in the Morien syncline.

The western continuation of the seam passes under the water of Indian bay, being only 1 ft. 8 in. in thickness at the outcrop in the shore cliff. It thickens progressively going eastward, ranging from 2 ft. 8 in. to 5 ft. where it has been worked in collieries Nos. 10, 11 and 24 of the Dominion Coal Company.

West of the Bridgeport anticline, the Emery seam apparently continues slightly in excess of 2 ft. in thickness and has not been worked. It thins approaching Sydney harbour and is not represented by any workable seam on the north side of the harbour.

No immediate undersea working of the Emery seam is probable, although No. 24 colliery of the Dominion Coal Company, a slope mine on the outcrop, is driving seawards; the face of the deeps being about half a mile from the shore-line, at the end of 1926.

It is probable that the Emery seam is workable in the submarine over a shore frontage of 3 to 4 miles in the Glace Bay district. Recent borings indicate that the Emery is of good quality over the area worked in the overlying Phalen in the No. 2 and 4 collieries, and of a thickness exceeding 3 ft. 6 in.

The Gardiner seam lies 427 feet below the Emery in the Glace Bay district. It has been mined only in a limited way, but was reached in November, 1926, by a shaft 560 feet deep, situated approximately $1\frac{1}{2}$ miles to the full dip of the outcropping, the seam, where proved, being 3 ft. 8 in. thick.

For economic purposes the Gardiner seam seems to be present only in the Glace Bay area. Its supposed equivalent in the Morien syncline, the Long Beach seam, is thin and dirty where tested. No workable equivalent to the Gardiner is known west of the Bridgeport anticline.

Below the Gardiner seam, in the Glace Bay area, at a depth of 180 feet, are found two seams, the upper 3 ft. and the lower 3 ft. 6 in. thick, separated by 35 feet of strata, and known locally as *Upper and Lower Lorway*. They are supposed to be identical with the lowest workable seam in the productive coal-measures, known to the east of the Glace Bay area, and west of the Bridgeport anticline in the Waterford district, as the 'Mullins' seam. The seam has not been worked. Near Low point in the Waterford district there is an area of this seam that is 6 feet in thickness.

Below the Mullins seam, and intervening between it and the *Tracey seam*, the lowest seam known in the coalfield, there is, very approximately, 1,900 feet of barren strata characterized by cross-bedding, and containing thin unworkable coal-seams.

With broad reference to the seams in the Sydney coalfield lying below the Phalen seam, and not yet mined under the sea, the following considerations suggest themselves.

The officers of the Geological Survey of Canada, namely, Robb and Fletcher in 1874 and later revisions, and Hayes and Bell in recent years, agree in placing the Emery seam at the contact of — to use the older nomenclature — the 'Coal Measure' and the 'Millstone grit'. The more recent studies of Hayes and Bell select a point 20 feet above the horizon of the Emery, marked by the first conspicuous entrance of anthracomya shells, as the base of the upper and most economically important series of coal-seams, so that, both on lithological and palæontological grounds, there is agreement that, subsequent to the deposition of the sediments that compose the roof of the Emery seam, there followed a long period with conditions favourable to the formation of coal seams extending over the whole coalfield, as we now know it.

The coal seams that exist below this zone of anthracomya occurrence appear to be of economic value under present conditions only in the central area of the coalfield, with exception of local thickening of the Mullins seam at Waterford and its probable submarine extension.

Shale zones, with associated thin coal-seams existing in the central and eastern part of the field, merge into massive sandstones towards the west and northwest. This sandstone mass, over 3,500 feet thick and barren of coal-seams, is stated by Hayes and Bell to be the equivalent of the horizon below the Emery, and from its presence is inferred "active transportation of sediment from a general northwestern direction" which has had the result that "as this source of supply is approached, strata, which elsewhere consist of rythmic alternations of coarser and finer clastics accompanied by numerous coal horizons, pass laterally into thick barren sandstones and grits."

These governing conditions of deposition suggest that undersea mining in the Sydney coalfield below the horizon of the Phalen seam will develop importantly only in the Glace Bay area, and that, chiefly, from the important vantage-point of Table head, where the Nos. 2 and 9 shaft is sunk to the Phalen seam. At this point the Lower Lorway seam, or the Mullins, should lie at between 1,600 and 1,700 feet from the surface, a depth that would permit only a limited advance into the submarine area, with favourable conditions of thickness and quality demonstrated to exist. The character of the seams below the Phalen has not been proved in this strategic vicinity, and their suitability for mining in the submarine area will require to be tested under the sea by prospecting from the floor of the Phalen seam.

The Tracey seam, lying at a depth (calculated on uncertain data) of 3,500 feet at Table head, is too deep to be mined under the sea, except in the Morien area as further mentioned.

Seams in the Sydney Field lying above the Hub Seam, and not mined:

Seams lying higher in the strata-column than the Hub seam are of little present economic importance, as their outcroppings are almost entirely submarine.

From the rapidly eroding shore-projections of Low point, Cranberry head, and point Aconi, however, the future may see very important winning of the upper seams, namely: Lloyd's Cove or Bonar Seams, and Point Aconi seam.—Correlation of the Lloyd's Cove seam is uncertain. The Geological Survey officers consider it to be the equivalent of the Bonar seams on Point Aconi, and one of the two seams lying above the Barrasois (or Hub) at Low point(1). Earlier correlations suggest the equivalency of the Lloyd's Cove seam with the Stubbert seam(2). Certainly the section of these two seams is similar.

If the Lloyd's Cove and Stubbert seams are correlated with each other, then there are two higher and untouched seams, namely:

Bonar seam which outcrops half-a-mile inland on point Aconi, and has an almost entirely denuded outcrop on Cranberry head, and is present at Low point; and Point Aconi seam, with a fragmentary outcropping only, at the tip of point Aconi (see accompanying photograph, Figure 8), the remainder of this seam being submarine.

The Lloyd's Cove seam has been mined intermittently since about 1870, but its quality is inferior to the largely-mined 'Main' or Harbour seam. The section of the seam is as follows:

Coal	2 ft.	0 in.
Clay		2 in.
Coal	,	6 in.
Clay		1 in.
Coal	3 ft.	6 in.
	6 ft.	3 in.

The seam outcrops on Cranberry head 1,800 feet from the shore-line. It has been worked in the No. 2 colliery of the Nova Scotia Steel & Coal Company to a distance of 5,100 feet from shore, passing under the sea with 110 feet of strata-cover, and having 501 feet of cover at point of maximum seaward penetration. This colliery was opened in 1902 and closed in 1904, being re-opened in 1908 and closed in 1915. The seam was also opened 2½ miles to the westward, at Bonar's head,

⁽¹⁾ Hayes and Bell, Geol. Survey of Canada, Memoir 133, page 101.

⁽²⁾ Dowling, D. B., Geo. Survey of Canada, Memoir 59, page 45; and Richard Brown, Coalfields of Cape Breton, page. 20.

in 1920, work being stopped in 1923, with the deeps just entering the submarine area. The seam was 5 ft. thick and the cover 139 feet at the shore-line.

Lying 184 feet above the Stubbert on point Aconi is the following assemblage of seams, namely:

Coal	2 ft.	6 inUpper Bonar seam.
Shale	5 ft. 1	11 in.
Coal		5 in with ½-inch parting.
Strata1	1 ft. 1	10 in.
Coal	4 ft.	1 inbanded.

Lying 208 feet above the Upper Bonar seam is the Point Aconi seam, 3 ft. 7 in. thick.

The strata overlying the Point Aconi seam are the highest in the coal-bearing series, and, until their disappearance under the sea, they continue to be of typical shales, including coalstreaks and fossil plants (1).

Because of the relatively inferior quality of the seams above the Harbour (Main) in the area west of Sydney harbour, they will not be extensively mined so long as coal of better quality is available, with cheaper extraction costs, in the central area of the Sydney coalfield.

However, within a period of little over thirty years we have seen the exclusive mining of seams, having thick sections of clear coal, supplemented and replaced by mining of thinner seams, requiring to be cleaned for marketing. This evolution, well-known in all coalfields that have been extensively mined, will continue.

It may therefore be confidently anticipated that the very large tonnages of coal lying under the sea, over a shore frontage of eleven miles, from Sydney harbour to the Great Bras d'Or, will later be extensively mined. The prospecting so far carried out is little more than a reconnaisance survey. Large-scale development will require to be preceded by systematic diamond-drilling, and elucidation of the geological structure, especially in the narrow but puzzling interruption of the coal outcrops by the Little Bras d'Or channel. The difficulties of correlation here arise chiefly from lack of an accurate base-map.

⁽¹⁾ Hayes and Bell, "Southern part of the Sydney Coalfield".

Entrance to the submarine extension of the seams, if attempted by driving in the coal-seams themselves, will be under light cover, and considerable seaward-drivage will be necessary before actual mining of the coal can begin. While the indicated mode of attack will be by shaft and cross-measure tunnels, this will necessarily be preceded by experimental slopes in the seams, and provision may have to be made for entrance under less than the 100 feet of cover required by the Coal Mines Regulation Act by use of reinforced passages, such as are contemplated by the British Columbia Act.

In regard to the Point Aconi location, which is the most interesting one for a future large-scale undersea colliery, it may be noted that the now completely-severed tip of the Point is shown on Richard Brown's map of 1870 as joined to the mainland of Boularderie island. The detached portion is a thin cliff of lower surface altitude than the mainland (¹). The configuration of the existing termination of the mainland is such that it also will eventually be broached by the sea, and severed, forming another island (See Figure 7).



Figure 7.—Point Aconi. Showing severed portion and progress of new breach by sea. Note abandoned road in foreground cut off by encroachment of the sea.

(Photograph by Douglas W. Johnson, 1923)

⁽¹⁾ Douglas W. Johnson. "New England — Acadian Shoreline", page 322

Reserves of Coal in the Submarine Area of the Sydney Field

There have been a number of attempts to estimate the coal content of the submarine area of the Sydney coalfield.

The writer will not attempt another estimate, as it is not in his opinion possible to calculate the reserve of 'recoverable' coal, because of the many separate but nevertheless interdependently-acting factors; such, for example, as quality and accessibility of the coal, and markets, all factors with a value varying with the date of mining.

It may be stated that the existence of a substantial reserve of recoverable coal is at this date more definitely established than at any previous date, and there is now sufficient coal virtually proved by existing workings to maintain the present rate of production for 200 years.

As there is no ascertainable knowledge of the un-mined seaward-coal, except that obtained by actual advance of workings, it is not possible to make a more accurate estimate than to state the balance of probability is that the coal-seams extend as far seawards as it will be possible to mine them.

Composition of Strata below the Sea-floor

The following table giving the composition of the strata overlying the Hub seam in the Glace Bay area — the highest seam exposed on land — shows that there is much uniformity in the nature of the sediments.

Composition of Strata overlying Coal Seams under the Sea in the Glace Bay Area of the Sydney Coalfield

Arranged from Report of the Geological Survey 1873-4, page 178

Arg. Aren.	Red + Green Maris	stone		Under. Clay	Lime-	Black Shale	Coal
42% 42%	3.4%	43.2%	106.6	3%	_	3%	0 9%
46 2%			Hub Seam	,			
28.47 28.57	4.1%	25%	339' // "	92%	2.4%	1.2%	1.4%
57%			Harbor Sea	197			
33.5% 15%		36.77	363' 8	8.7%	3.8%	1.1%	3%
48%			Phalen Se	Q m			

Below the Phalen seam, the percentage of sandstones and calcareous sediments increases, but, from the standpoint of undersea mining, the strata-cover above the Hub seam is most important. This contains much sandstone, one massive layer 25 feet above the seam having furnished good buildingstone. Above this is a 3-foot bed of underclay below a thin seam of coal. Between the Hub seam and the Harbour seam are ten occurrences of clay-shales, classed as 'underclays'.

The only indication of the nature of the strata overlying the Hub seam out at sea is the section of the Point Aconi area, where in the 409 feet intervening between the Point Aconi seam and the Hub seam (Stubbert), the strata has the following composition:

Various shales	per cent
Sandstones	44 '
Coal	"
Underclay 2	66
Limestone conglomerate	44

The occurrence should be noted of a bed of underclay, 8 ft. 1 in. thick, underlying the Lower Bonar seam, 190 feet above the Hub seam (Stubbert).

The large proportion of shales in this section, with the occurrence of one thick underclay, indicates a more elastic and less-permeable cover than the immediate roof of the Hub seam, and while there is great dissimilarity between the character of the strata with relation to included coal-seams in different parts of the Sydney field, the composition of the Point Aconi strata-section suggests that the measures forming the sea-floor, and those immediately below it, are of similar character over the whole undersea field.

Coast Erosion

The seaward inclination of the parallel-bedded shales, clays, and sandstones of the Coal Measures in Nova Scotia, aided by the action of frost, hastens the ordinary effect of wave erosion, so that retreat of the coast-line proceeds rapidly, especially in Cape Breton island. The 25-fathom depth-line in the Sydney coalfield runs at a distance from shore of from seven to ten miles, and the intervening distance is part of a shelf of progressive marine-denudation that extends far out under the sea (1).

⁽¹⁾ Trans. Inst. Min. Eng., 1903; Discussion by H. S. Poole of paper by T. E. Forster.

This matter of erosion of the coast-line is one of some interest in any long-range consideration of undersea mining in Cape Breton island, because both in the Sydney field and in the separated fields on the western coast, there has been within historic times disappearance of the outcroppings of coal seams through encroachment of the sea.



Figure 8.—Point Aconi. A closer view of the point shown in Figure 7 At the extreme point, in front of and below the figure standing on the cliff-edge, can be seen the remnant of the Point Aconi Seam with a slight capping of shales.

(Photo by Douglas W. Johnson, 1923)

The accompanying photographs (Figures 7 and 8) illustrate the extent of the erosion and the fragmentary character of some of the visible coal outcroppings.

In 1870, the late Richard Brown urged upon the Nova Scotia government the necessity to ascertain the exact position and value of such coal-seams as remained above low-tide mark in the Inverness field, and the advice is not less applicable at this date.

It has been necessary in one or two instances to build dams and concrete supports to strengthen portions of mine workings that have approached dangerously near to shore outcroppings of coal seams.

A customary method of approach in the early days of coal-mining in the Sydney field, where coal seams out-cropped at the shore-line, was to "drive a level from the water's edge along the strike of the seam." (1) It is natural that the seams should have been first attacked where they showed so plainly as at the shore-line.

Richard Brown (2) estimated the erosion of the shore at the rate of 5 inches per year, and mentions slips of the shore cliffs at several points, especially on the southeast shore of Sydney harbour, where the measures are steeply inclined. He instances a slip half a mile long, two hundred yards wide, and 20 yards high, that slid off the upper surface of a coal-seam. Photographs accompanying this paper show more recent evidences of erosion in this vicinity (See Figure 9).



Figure 9.—Shore cliff on southeast side of Sydney Harbour entrance, in the vicinity of the slip mentioned by Richard Brown in 1870.

Shows road abandoned, and eroded by the sea. The faint shore in the upper left-hand corner is Cranberry Head and the Princess Colliery.

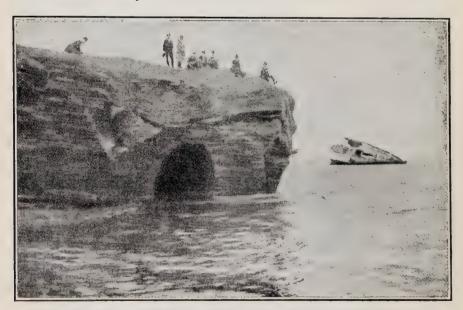
(Photograph taken in 1926 by Mr. B. A. L. Huntsman)

⁽¹⁾ Coalfields of Cape Breton, by Richard Brown; pages 87 and 89.
(2) Coalfields of Cape Breton, by Richard Brown; pages 12-13.

The Dominion Coal Company has made observations over the past five years which indicate erosion at the rate of 10³/₄ inches annually, and the evidence of old plans, over a period of 25 years, indicates a rate of 9 inches annually. The correctness of the old plans is open to question.

At the outcrop of the Emery seam, at Dominion, an iron post set on a coal-lease line was described as being in 1893 a distance of 85 feet from the shore-cliff. In 1921 the post was at the edge of the cliff, and "toppled" over in that year. This would indicate erosion at this point of three feet per year, which is not representative (1). The rate of erosion varies, of course, with local conditions. The series of photographs accompanying the text shows the wasting of a spur of rock near Glace bay, the formation of a stack, and its almost entire destruction within a period of 25 years (See Figure 10, six views).

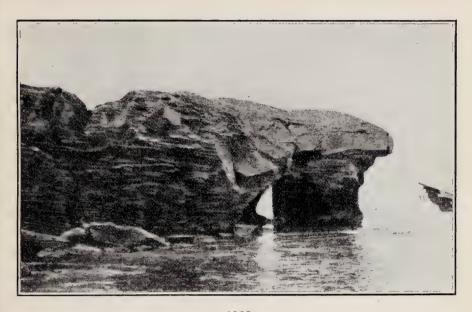
It is probable that the annual average recession of the coast-line due to erosion is nearer the larger rate of 9 inches ascertained by recent observations than the smaller rate of 5 inches recorded by Richard Brown.



(Original photograph from Mr. Stuart McCawley, Glace Bay, N.S.)

Figure 10.—See continuation on pp. 47-49.

⁽¹⁾ These particulars were furnished by Mr. Alex. Hay, Asst. Mining Engineer of the Dominion Coal Company.



1908



Figure 10.—See continuation on pp. 48-49.



1918



1920

Figure 10.—See continuation on p. 49.



1926

Figure 10.—Sandstone Spur near Table Head, Glace Bay, N.S. Showing almost complete erosion in 26 years.

(Last four photographs taken by Mr. B. A. L. Huntsman)

The erosion of the Carboniferous strata in the Sydney district is very fully dealt with in a recent work by Douglas Johnson (1). His conclusion is that

"despite the rapid incision of the waves, the total amount of erosion accomplished at the present level does not appear to be very great.... the total breadth of the zone thus lost to the sea is apparently to be measured in a few thousands of feet rather than in miles.

"Accordingly we are led to conclude that the time during which the sea has operated at the present level cannot have been great, probably a matter of some few thousand years at most (2).

Mr. Johnson adduces as evidence of the recent age of the present erosion-level of the sea the scarcity of cliffs of diminishing altitude.

^{(1) &}quot;The New England-Acadian Shoreline", by Douglas Johnson, 1925; pages 335-336. John Wiley & Sons, New York.

⁽²⁾ Compare Richard Brown's estimate of 25,000 years "since the coal country emerged from the water of the Atlantic". Coalfields of Cape Breton, page 12.

Inbreaks of the Sea

The recorded instances of inbreak of the sea into working collieries are not sufficiently numerous or important to indicate that mining of coal under the sea bed is unduly hazardous, if reasonable precautions are used.

The Workington disaster, previously noted, which, Mr. T. E. Forster states, seems "to have pursued the subject of undersea coal-mining all over the world", was not so much an accident as a result of gross carelessness (1).

The inbreak at Mabou was of similar character, and the flooding of old mines mentioned as having occurred in Chile was a result of unskilful mining.

With regard to the flooding of the Port Hood mine, the writer believes that the whole story has not transpired, but the incident, unless further explained, is the most disturbing yet encountered in undersea mining of coal.

It is significant that in the Workington, the Mabou, and the Chilean instance, water was admitted, not through the shore opening, but by workings carelessly driven back in the direction of the shore and towards the rise. The question of roof support hardly enters into the cause of these floodings. If the Port Hood mine was flooded through a vertical break. it is a unique instance. It is the writer's belief that it was not so flooded (2).

Mine Fires

The water of the sea has been admitted into mine workings to drown out fires on two occasions in the Sydney field.

A fire in the Phalen Seam workings in Dominion No. 1 colliery was extinguished in this manner in 1903, (3)

Another fire in the Hub seam was dealt with in the same manner in the winter of 1906-7. In both instances use was made of old shore-workings. Both these fires occurred from the careless use of naked lights.

 ⁽¹⁾ Trans. Inst. M. E., 1903; also Vol. xxiii, p. 663, 1902.
 (2) See page 75.
 (3) "The Fire in Dominion No. 1 Colliery", by Shirley Davidson and Norman MacKenzie; Proc. Can. Soc. Civil Eng., 1906.

In the last-named instance, the fire was at the shaftbottom, or virtually the highest point in the mine as then worked. The water was first admitted through opening-up of an old shore-level near high-water mark. The inflow of water occurred only at high-tide, and free entrance was hindered by silting up of the opening by gravel at each high tide. It was impossible to clear away the gravel at low tide because of the emission of carbon dioxide gas from the opening in such quantities as to extinguish safety-lamps on the beach. The opening was, moreover, situated under a dangerous overhang of cliff weighted with icicles. It was necessary to sink a shaft, placed slightly back from the edge of the cliff, to the coal-seam, and to drive out in the seam to a point between high and low-water mark, where a communication was made with the sea. This opening was later closed by a concrete dam. The unwatering of the mine occupied ten months.

The mine was flooded in January and much 'slob' or 'lolly' ice accompanied the sea-water, leaving a frozen saline-deposit resembling coarse sea-salt, and of considerable thickness, along the rim of the water as it receded. The deposit had absorbed an appreciable quantity of carbon-monoxide, and headache and nausea was experienced by the workmen in clearing a way through.

A fire occurred at the foot of the Princess Colliery deeps in 1908 (1), which was successfully fought with the aid of oxygen breathing-apparatus; but apart from the circumstance that the fire occurred at a point two miles from shore it does not call for comment. This fire originated from a 'hung' shot.

Equipment in the Sydney Coalfield

It is not proposed to describe the equipment, or the details of mining practice, except where these are modified to meet submarine conditions.

Two factors are outstanding, namely, the extraordinary importance of ventilation and haulage when compared with land-mining standards. Both are problems arising out of distance.

^{(1) &}quot;The use of Breathing Apparatus at a Mine Fire in Cape Breton" by Gray and McMahon; Trans. Inst. Mining Eng., 1909.

Appendix 'E' gives details of ventilation practice in submarine mines in the Sydney field. When it is realized that all the mines named are expected to proceed seawards at least as far again as they have already gone, the necessity for airways of sufficiently ample area to restrain water-gauges within reasonable limits is clear.

Endless haulages appear better suited to submarine conditions at the present stage of development, and have replaced trip haulages in a number of instances. The method of haulage for remoter undersea-mining remains to be developed.

The curved level-line of the No. 1B main-haulage is a condition suitable to electric-trolley haulage, and the command over the dip coal given by the curve of the main-haulage road around the anticlinal dome has been taken advantage of to instal powerful 'drop-hoists' for dip-haulage. These hoists, both electrically-operated and air-operated, have been designed with regenerative devices, the weight of the descending cars being utilized to generate current in one instance and to compress air in another (1).

Electric power is used in the submarine mines for haulage and pumping, and in one instance for auxiliary ventilation, but is not used at or near the coal-face, where compressed air is employed.

As elsewhere suggested, the more extended use of electric power in undersea operations in the Sydney field is probable, indeed inescapable. If the use of electricity at the face should continue to be regarded as inadmissible, the compression of air at considerable distances 'inbye' will probably develop importantly.

Pumping is not relatively important in undersea operations at depth, except as it may be necessary to deal with land-measures drainage at the shore. Deep submarine workings are, in the Sydney field, as elsewhere under similar conditions of depth, so dry as to raise the necessity for dealing with coal-dust accumulations. Except for the possibility of inflow from the sea, the necessity to deal with mine water is much

^{(1) &}quot;Hoist installation at No. 1B Colliery", by H. V. Haight, Can. Min. Jour., June 18th, 1926, p. 617.

less onerous than in operations under water-bearing strata, such as the Magnesian Limestone in the north of England and the water-bearing sands of the Lower Oolite in the Kent coalfield.

Riding of men to and from the working sections is general in the submarine collieries, and in some instances a haulage-road and hoisting equipment is provided exclusively for man-haulage.

THE SPECIAL PRACTICE OF UNDERSEA COAL-MINING

The possibility of uncontrollable inbreak of the sea consequent upon rupture of the sea-floor by removal of coal is the chief factor in deciding methods of coal mining under the sea. Whatever differences exist between mining coal under the sea and under the land originate in this dominating consideration.

Distortion of the strata is especially to be guarded against. In this connection, the working of several coal-seams in the same area requires careful planning to avoid the superposing of solid coal, as, for example, barriers, and pillars of large area, which may cross or be precisely superposed and be surrounded by worked-out areas, in wo or more superimposed seams. Stowing of the goaf against such large permanent pillars would minimise the strata strains.

This possibility of cumulative superimposition, with consequent distortion of the measures, was one reason counselling abandonment of the panel system previously prescribed by the Coal-Mines Regulation Act in Nova Scotia.

Where the mining of one seam only under the sea is in question, or possibly two seams, the ideal method of extraction is entire removal of the coal by the longwall method, with careful packing of the waste; but use of this method is only possible where roof and floor conditions favour longwall, in seams of suitable thickness, and where packing material is available.

The seams so-far mined in the submarine area of the Sydney field have exceeded six feet in thickness of clear coal without parting, and the extension of workings from land areas under the sea by use of the longwall method was not attempted — being, indeed, at the dates most of these extensions took place, forbidden by the Coal-Mines Regulation Act.

The use of sand and similar materials to pack the waste by employment of the hydraulic-flushing method has been suggested, and while this method may yet be utilized for recovery of pillar-coal now standing in worked-over areas on land, where it is possible to convey the packing material by shallow boreholes, the writer has been unable to find any instances of the use of this method under the sea. There is available sand conveniently situated in the vicinity of the collieries in the Sydney field, which could be dredged and landed close to shore collieries at moderate expense, but the underground conveyance of sand or other packing material, such as engine-ashes, by water for long distances seawards would be very difficult, because of the gentle inclination of the seams under the sea. The return pumping of the water for long distances uphill is another difficulty.

Very extensive areas of pillar-coal exist in the Sydney field between the shore and the line of 800 feet to 1,000 feet of solid strata-cover below the sea-bottom where commencement of entire removal of the coal has been considered permissible, and it may at some future date be found possible to use hydraulicly-transported packing material to permit extraction of these pillar areas, as has been suggested by Mr. Walter Herd (1).

The order in which superimposed seams should be worked is a matter that has been much debated in regard to ordinary practice in land mining, but it is a point of especial importance in undersea mining. There does not appear to be any agreement as to the most advantageous order of extraction in underseamining practice outside of Nova Scotia. The choice of order seems to have been decided by immediate economic considerations, as has been the case in Nova Scotia.

The writer believes that if statement of a broad rule is possible in undersea operations, seams should be worked in descending order, and this was the order in which the seams were developed under the sea in the Glace Bay area from Table head. Later, as has been explained, the working of the uppermost seam, the 'Hub', was given up because of high

^{(1) &}quot;The suggested application of Hydraulic Stowing to undersea coalworkings, with special reference to the Sydney Coalfield", by Walter Herd: Trans. Min. Soc. of Nova Scotia, 1920.

production-costs. Mining in the lowest seam worked (Phalen), 7 ft. 6 in. in thickness, disturbed main roads in the Harbour seam workings, 400 feet directly above.

It was desirable to commence the extraction of pillars in the lowest seam, where only had workings attained a depth permitting this. The topmost seam is full of water, due to cessation of pumping, and in order to ensure an adequate margin of safety it was decided to treat the position of the topmost seam as equivalent to sea-bottom level. This decision increased the depth at which total extraction of the lowest seam (Phalen) was permissible to 1,200 ft., instead of 800 ft., and, of course, very greatly increased the distance from shore to the point where total extraction was permissible in the middle seam (Harbour). In these circumstances it was decided to suspend mining in the middle seam, and rapidly advance the workings in the lowest seam, proceeding with pillar removal at the earliest opportunity. After removal of pillars in the lowest seam, a sufficient lapse of time will be allowed for settlement of the intervening strata, after which it is intended to resume the advance of the workings in the middle seam, over territory in which subsidence has taken place and which, it is anticipated, will be sufficiently 'set' to remain without further movement. Thereafter, the advance of the faces in the two seams will be arranged to maintain the relative position desired.

Mining in this middle seam (Harbour) has been suspended since 1921 and will not in all probability be resumed until 1931.

The Emery seam, probably slightly under 4 feet in thickness, lies about 150 feet below the Phalen seam, and if worked in the Glace Bay submarine area will be the fourth in descending order and the lowest seam to be worked. If this seam is mined by the longwall method, for the application of which it is suitable, and the waste is carefully packed, a minimum disturbance of the workings in the three superposed seams should take place. It is not certain, however, that this seam is present in sufficient thickness or continuity to be workable under the western portion of the Glace Bay submarine area.

In the Waterford area, as elsewhere noted, the matter of the order of working seams in the undersea area has arisen, the question of possible disturbance of upper-seam workings by advance of workings in a lower seam being at this time under advisement.

The importance of the effect of order of extraction upon the future of the Sydney field is very large, because in the centre of the field there is 40 feet of coal included within a strata column of less than 1,000 feet. Six of these seams should be workable now or at some future date.

Nevertheless, a comparison of practice elsewhere with the record of experience so far obtained in the Sydney field, does not permit one to draw any hard and fast conclusions upon the best order of extraction. Obviously, questions of spacing and thickness of the seams, and the nature of the associated strata, must be considered along with questions of marketability of coal, and a decision made upon these local considerations. The condition most to be guarded against is distortion rather than subsidence, or perhaps it may be said that the condition most likely to cause rupture of the strata is unequal subsidence.

The detection of points of weakness in the strata-column and adoption of precautions suited to these weaknesses is required in undersea mining. Among these weaknesses may be cited faults; rocks of porous or jointed character, as some sandstone beds; the absence of underclays; alluvial or sand pockets occupying depressions in the sea-floor; and sudden drops in the sea-floor, such as may mark ancient beach-lines (1).

Considering the duration of coal-mining in the Sydney field, relatively little experience has accumulated with reference to subsidence of the surface consequent upon working of the coal-seams. In very few instances has more than one seam been extracted under populated areas, where subsidence damage attracted attention because it resulted in financial loss.

The most important instance is the working of the Harbour and Phalen seams under the town of Glace Bay. Subsidence in this locality became marked when the Phalen seam was worked beneath an area of the Harbour seam having standing pillars of old date. The draining of the pillared waste of the

⁽¹⁾ See discussion by H. S. Poole, Trans. Inst. Min. Eng., 1903.

Harbour seam removed one source of support, and, judging by the extent of the subsidence, the pillars, presumably of inadequate size, suffered extensive crushing.

But neither at Glace Bay nor elsewhere in the field has there been *complete* extraction of two seams in superposed areas. Reference to the map, showing the successive character of the outcroppings, will disclose that only in a very limited land area, having Table head, Glace bay, as its 'hub', does a superimposition occur of all the coal seams so far worked.

As all mining in this district, with negligible exception, followed the dip of the seams downwards from their outcroppings, employing pillar-and-room methods with variations in the size of the pillars left standing, and with unsystematic and incomplete recovery of pillars; and as the depth of mining has everywhere in this area been less than 800 feet from the surface; the observations possible to be made with regard to surface subsidence, both as to superficial extent and vertical drop of the disturbed measures, have been limited. Neither has there been any pressing necessity to make such observations, because of the largely unoccupied nature of the mining districts, and the absence of stone structures.

There is a lack, therefore, of the precise data upon which to formulate theories of subsidence factors under the adjoining submarine tract, in which, for economic reasons, it is desirable, and probably necessary, to effect a relatively complete extraction of a number of superimposed coal-seams at greater depth than practised under the land area, where the preliminary experience has been gained.

No problem of coal-mining is more dependent upon local conditions than that of subsidence consequent upon removal of the coal seam itself. In this connection there may fittingly be quoted the conclusion of a German student of coal-mining subsidence, who states that:

"after having striven to throw light upon the unknown nature of the movements of great rock-masses set up underground by coal-mining, and their effect above-ground, my conclusion is that it is quite impossible to arrive at exact data as to the boundaries of an anticipated subsidence area, or with regard to the vertical extent of the sinking.... it is necessary to warn against the mechanical use of formulæ. The problems of subsidence can never be brought within the limits of a rigid system." (1)

^{(1) &}quot;Earth movements in coal mining and their influence upon the Surface", by A. H. Goldreich; Berlin, 1926. ("Die Bodenbewegungen im Kohlenrevier und deren Einfluss auf die Tagesoberfläche.")

The writer ventures the opinion that the more recent practice of submarine mining in the Sydney field has erred, if at all, on the side of safety and conservatism, and if later developments should show this to be the case, the unavailability to the mining engineers of today of reliable local data upon subsidence, and the necessity to accumulate such data by empirical methods, is self-evident.

Nevertheless, as elsewhere suggested, the subject is one now calling for greater study than has hitherto been accorded it in Nova Scotia.

In the Sydney field, faults are fortunately rare. A reverse fault in the Caledonia submarine area in the Phalen seam (¹), and the reverse fault elsewhere mentioned as being present in the submarine area of the Hub seam, are the only instances noted by the writer in the Sydney submarine area that have been encountered in working. In each case the dislocation was only equal to the thickness of the seam. The undisturbed nature of the coal measures in the Sydney field is a very favouring condition, and is in marked contrast to other instances of undersea mining herein mentioned, notably in Chile, in the Whitehaven coalfield, and in Japan.

The existence of alluvial pockets and holes in the sea-floor, to which Mr. Atkinson's paper drew attention in New South Wales, is not a factor in the Sydney field, where, as elsewhere noted, the sea-floor is chiefly the scoured rock of a shelf of marine denudation of very recent excavation. There is, of course, no over-clay.

The possibility of entrance of water through the interbedded laminations of inclined strata outcropping under the sea should be noted (2). Richard Brown notes the entrance of sea-water into exploring levels driven under the sea in the Harbour (Main) seam through a thickness of 300 feet of strata "which it was necessary to shut off by strong framed dams" (3). These levels were being driven towards the outcropping of the seam under the water.

^{(1) &}quot;Description of Caledonia Colliery", by Eugene P. Cowles; Trans. Can. Min. Inst., 1909, page 529.

⁽²⁾ See reference to flooding of Port Hood colliery, page 74 of this paper (3) "Coalfields of Cape Breton", page 131.

Mine Plans and Surveys

Very accurate surveys and careful notation on the mine plans of the contour of the sea-floor, as established by soundings, are necessary in undersea mining. In the Sydney field, soundings have been taken for a number of years, and the sea-floor over almost the entire area under present and prospective development has been contoured (1).

The superposing of plotted workings where several seams are worked in the same area, which is a necessity in the work of the mining engineer, produces a complex arrangement, difficult to grasp, and tiresome for the draftsman to detail on paper. A possible method would be to paint a skeleton plan of workings in superposed seams upon plate glass. The plates could be arranged to correct vertical scale in a rack having also glass sides on which the section could be drawn, so that the complete arrangement would reproduce in miniature a transparent cubical block of the measures (2).

METHODS OF EXTRACTION

The pillar-and-room method of extraction has always been preferred in Nova Scotia. This has been the case because thick clean seams were for many years mined to crop openings at shallow depths, and the traditional method has survived, being today employed to win thinner seams at much greater depths, conditions under which there is good reason to believe longwall extraction would give better results.

Longwall mining is not a new method in Nova Scotia. Richard H. Brown mentions successful longwall mining in the Princess Colliery submarine area in 1899. The Gardiner seam was mined by longwall faces in 1892. A completely developed and entirely successful longwall operation was carried on in the Emery seam from 1908 until the early years of the war. A successful submarine longwall operation at Joggins, N.S., is elsewhere noted.

⁽¹⁾ For a detailed account of the survey practice of the Dominion Coal Co., see paper by Alex. Hay, Trans. Min. Soc. of Nova Scotia, 1925.

⁽²⁾ See "Coalfields and Coal Industry of Eastern Canada"; Bulletin 14, Mines Branch, Dept. of Mines, Ottawa, page 35.

The writer believes that longwall mining in the Sydney field has failed to progress as it has done in other fields because of conservatism and unfamiliarity of local miners with the method, and through insufficient numbers of miners trained to longwall work locally available, and not to any inherent inapplicability of the method to local conditions (1).

In discussing the merits of the longwall method as applied to local conditions, J. J. McDougall states that

"while the longwall system is usually adopted in these seams, it is also found to be advantageous in thick seams where the overbearing strata is so heavy that the maintenance of roadways in rooms, and the extraction of pillars, add materially to the cost of production (2)."

The President of our Society in 1921 stated:

"Experience in pillar drawing proves conclusively that under such conditions as obtain in Princess colliery, with a cover of 1,000 feet or over and a roof of the nature so far found, some method must be adopted whereby pillars can be removed simultaneously with the advance of the workings." (3)

This is precisely the condition that will be encountered in every seam worked at depth in the submarine area. The conditions are in fact that partial extraction of a coal-seam has under these circumstances become impracticable, and all or none of the seam must be extracted. As longwall achieves an equal subsidence (if there is ignored for purposes of this argument the vertical extent of subsidence), whereas under the conditions in question the leaving of pillars causes an unequal subsidence, longwall extraction would seem not only preferable but without alternative.

Very considerable progress is now being made in adapting the longwall method to mining in the Sydney field.

At the close of 1926, the percentage of production from longwall faces in new work in submarine territory was as follows:

	Per cent
Sydney Mines Collieries	40
Waterford area	
Glace Bay area	

^{(1) &}quot;Coalfields and Coal Industry of Nova Scotia"; Mines Branch, Dept. of Mines, Bull. No. 14, page 39.

^{(2) &}quot;Longwall operations, Sydney Mines, N.S.", by J. J. McDougall; Min. Soc. of N.S., 1925.

^{(3) &}quot;Notes on mining in Submarine Areas", by A. S. McNeil; Trans. Min. Soc. N.S., 1921, p. 287.

Some of the factors adding to the cost of production and the saleability of coal won from submarine areas at this time may be noted in this connection.

Distance from shore.—Longwall mining, by extracting the whole coal-seam in one operation, advances slowly by comparison with pillar-and-room, and covers a smaller area. As compared with pillar-and-room, the use of longwall delays the advance of workings seawards.

Ventilation difficulties.—The difficulties of ventilating the 'broken' area of a room-and-pillar mine, compared with the comparative ease of ventilating the more compact and 'tighter' airways of a longwall operation, require no emphasis.

Quality of coal.—Probably the chief drawback to the marketability of the coals from the Sydney field is smallness of texture at destination. Some of this condition is due to the special conditions of transportation, but more is due to the large quantities of powder used in blasting out a friable coal during mining. The large size and unshattered structure of coal won by longwall faces, using little or no explosive in getting, will be admitted.

Depth of mining.—This feature has been dealt with. It is possibly trite to add that all future coal in the Sydney field must be won from greater depths than have to date been the rule.

While it is not desired to convey the impression that longwall methods are a panacea, or to imply that they are generally applicable, it is suggested that longwall could very profitably have been more extensively used in the submarine area of the Sydney field than it has been, and that its wider application is likely in the near future — cannot indeed be much longer delayed.

CONDITIONS OF DEPOSITION — SYDNEY COALFIELD

In order to assess what the future of undersea mining in the Sydney coalfield may be, it is necessary to understand the conditions under which the coal-seams and the associated sediments were laid down. If, as supposed by Hayes and Bell, deposition took place on a subsiding flood-plain, then the beds first laid down would be of smaller superficial extent, and confined to the centre of the basin of deposition. The beds last laid down would extend over the maximum undenuded extent of the young coalfield. It follows also that in the centre of basin of deposition (which it is suggested lies roughly along the seaward course of the Bridgeport anticline) a greater thickness of sediments, with numerous included coal-seams, will be found; and that, as the rim of the depression in which the sediments accumulated is approached, fewer coal-seams, and these of smaller thickness, exist. It follows further that the seams of the central area will thin out and be invaded by dirt bands as they approach the delimiting slopes of this depression, and that coarser sediments and impurities in the coal-seams will mark the approach to the source of the sediments. The condition is one of transgressive overlap, and is diagrammatically shown by Figure 11.

This reasoning is based upon actual conditions in the Sydney field. It explains the deterioration of the seams of the Glace Bay basin as they are followed westward, the paucity of economically workable seams in Sydney Mines-Bras d'Or area below the horizon of the Harbour, and the impoverishment of this seam itself west of the Little Bras d'Or entrance.

A puzzling feature of the geological structure is the Cape Percy anticline, separating the Glace Bay area from its outlier in the Morien area. The shallow corrugations of the field are evidently of date subsequent to the deposition of the coal measures, and do not affect the thickness or quality of the coal-seams. There are some indications that where the anticlinal ridge of cape Percy now exists there may have been a contemporary spur of high ground projecting into the original basin of deposition, causing a shallowing of the swamp and local coarsening of the sediments, because there is definite deterioration of the coal-seams (the outcrops or horizons of which parallel the existing ridge) as they approach the ridge on both sides of the anticline, suggesting deposition on the side of a hill. This contamination of the coal-seams must have been the result of conditions anterior to the denudation of the seams along the anticlinal crest.

The lines of folding in the Sydney field are parallel to the general Appalachian folding in Nova Scotia and Newfoundland, but the Cape Percy anticline is almost east and west in course, and diminishes landwards, whereas the other folds diminish seawards and have a course about northeast by southwest.

A significant condition is the presence of impurities in the seams on the Glace Bay side of the Cape Percy anticline, which is not associated with alteration of dip seawards, or variation in the thickness of the seams. The impurities which lessen going seawards - are not in the form of bands invading or splitting the seams, but the coal is higher in ash and of coarser texture, and the sulphur content increases. There is little doubt that the sediments and coal seams were synchronously laid down around the contemporaneous Cape Percy elevation, and may even have mantled it; and it seems probable that the coal seams curve round the seaward continuation of the existing ridge into the Morien area, as indicated on the map (Figure 2). The structure might be made clearer by careful observation of the dips and strikes on Flint islands, and the evidence yielded by a coal-seam, having apparently about the horizon of the Gardiner seam, that is exposed on one of them.

No really accurate data exist that will throw light on the character of the eastern portion of the coalfield. Exploration has been discouraged by the deterioration of the coal-seams in this direction, but it will be seen from the map that a large submarine coalfield is accessible from the northern shore of the Cape Percy headland. There is the possibility that the contaminated character of the seams may, as previously suggested, be associated with the existence of an elevation contemporaneous with the deposition of the coal-seams, and that this condition may be merely local. The conditions on the limited land-exposures of the Cape Percy headland are sufficiently unusual to suggest that the character of the seams seawards should not be assumed from the nature of the seams at the shore.

Distant 2½ miles from cape Percy are Flint islands, the larger island — on which a lighthouse is situated — being slightly less than a quarter-mile in length, apparently a remnant

of the Cape Percy ridge; and, judging from the geological ordnance-map — and not from personal observation — situated on the Glace Bay side of the anticline.

Great importance in regard to the economic value of the submerged eastern portion of the Sydney coalfield attaches to such geological evidence as may be preserved on Flint islands, or may be disclosed by borings there. Should exploration disclose workable coal-seams in the eastern portion of the field, it is possible Flint island may be used for future mining operations.

There is no evidence of a termination of the coalfield under the sea in this vicinity, nor does the contour of the sea-bed, as disclosed by soundings, suggest anything of this kind.

A similar lack of actual knowledge exists with regard to the western termination of the coalfield. Here there is fairly definite evidence that the field terminates against a seaward continuation of the Cape Dauphin fault, a condition suggested by the alignment and shape of the lower-Carboniferous outliers known as Bird islands. It is not known, however, whether the topography of the original basin of deposition allowed of maximum development of the coal-seams up to the line of the Bird islands, but these islands are composed of barren grits, a condition which suggests that such is not the case.

Splitting of the coal-seams is not uncommon in the Sydney field. Reference has been made to a splitting of the Hub seam by a shale inclusion, of undetermined extent, that may be lens-shaped. A parting in the Phalen seam encountered in mining the land area in the Glace Bay district reached at one point a thickness of 28 feet, but decreased on all sides, disappearing entirely in the undersea workings, being therefore a lens-shaped inclusion of local extent. This parting recurs on the Waterford shore, and, going westerly, fans out, the seam eventually being lost by invasions of shale bands. Similarly the Hub seam (if correlated with the Chapel Point series) splits up into a number of small seams in the Sydney Mines area, but these recombine and form a seam 8 ft. to 9 ft. thick still further west (1).

⁽¹⁾ Compare with conditions noted by Prof. P. F. Kendall, "On the splitting of coal-seams by partings of dirt"; Trans. Inst. Ming. Eng., Vol. LIV. 1917, p. 460.

The inference to be drawn from these instances is that the characteristics of a given seam in the Sydney field cannot be accurately gauged from a single proving. The writer suggests that the conditions of deposition indicate that seams which are split along the general line of the shore will tend to come together and develop their maximum thickness seawards, and more particularly in a general northeast direction.

The frequent recurrence of splitting of the coal-seams indicates that oscillations of level of the original plain of deposition were not confined to the definite emergences and submergences represented by existing coal-seams and thick strata-intervals, but that minor changes of level occurred during the accumulation period of the individual coal-seams.

The numerous dirt-partings found in the seams outcropping along the Cape Percy anticline suggest that oscillation was more frequent in this vicinity than elsewhere in the field (See Figure 11).

This feature may indicate a connection between changes of the flood-plain level and the sulphur content of the coalseams. Some students of sulphur occurrence in coal-seams associate it with deposition of the coal-substance in coastal marshes containing brackish water (1), suited to the activities of sulphur bacteria.

Dr. Reinhardt Thiessen, the outstanding authority on this matter, believes that organic sulphur in coal is derived chiefly from the original plant substance (2). It is, of course, possible that the character of the vegetation may have varied with brackishness of the swamp water, such brackishness in a coastal swamp tending to vary with changes of ground-level. Such a condition is not entirely inconsistent with the assumed fresh-water origin of the coalfield. It may have been an estuarine or delta deposit.

Very little massive marcasite, such as might infiltrate a coal-seam from outside sources, is present in the Sydney seams, the sulphur being present as very-finely disseminated pyrite and as combined sulphur (3).

⁽¹⁾ Pirsson & Schuchert, "Historical Geology", pp. 787-788.
(2) "Occurrence and origin of finely disseminated sulphur compounds in Coal"; Trans. Am. Inst. Min. Eng., vol. 53, 1919-20, pp. 913-926.
(3) "Treatise on Sedimentation", by Wm. H. Twenhofel; 1926, pp.333-

This matter is of first-rate economic importance, and while additional light on the origin of sulphur in coal may seem to have an academic value only, and an importance inferior to knowledge of some method of ridding coal and its products of sulphur, this does not lessen the desirability of careful research into the conditions of deposition favouring the presence of sulphur in an unprospectable area like the Sydney submarine coal-field, because it is conceivable that such research might lead, in the case of future mine-projections, to the avoidance of sulphur areas.

It is a matter of some interest that the sulphur content which marks certain coal-seams at their land-outcropping on the Cape Percy anticline diminishes seawards and *eastwards*, indicating the probability of purely local contamination associated (vaguely, it is freely admitted) with local changes of level contemporaneous with deposition.

In these observations, the writer is suggesting the necessity for research rather than attempting to formulate a theory, and if the discussion appear nebulous, it is not more so than our actual knowledge of the submarine field itself outside the limited area which has been mined.

GEOLOGICAL MAPS

The maps and memoirs of the Geological Survey of Canada which deal with the coalfields of Nova Scotia are of dates between 1870 and 1884, and are the foundation upon which the economic development of the coal deposits has been built. This pioneer labour is associated with the names of Richard Brown, Sir J. W. Dawson, Sir Wm. Logan, Hugh Fletcher, Edwin Gilpin, Henry S. Poole, and others of a generation that did solid and enduring work, and to whom we are deeply indebted.

In more recent years, with the exception of the work of Hayes and Bell, there has been no geological research at all comparable with that done in the 'seventies, and but meagre application of the tremendous accretion of knowledge that has been achieved in coal geology since those fruitful years, which were after all but the early beginnings of historical and applied geology.

Probably no persons interested in coal-mining in Nova Scotia are so conscious of the incomplete character of the geological maps of the coalfields as are the officers of the Geological Survey, whose known desire to be helpful is limited by meagre appropriations and an insufficient number of trained coal-geologists. The Province of Nova Scotia itself has not in recent years attempted any participation in geological research in regard to the coal and other mineral resources it owns, contrasting in this regard unfavourably with Alberta, British Columbia, and Ontario. It is fairly obvious that maps dated 1873 and 1884 cannot contain the information that has since been made available by fifty years of mining, and the illuminating character of the new map of the southern portion of the Sydney coalfield — a district in which mining has been negligible in extent — which resulted from the recent work of Hayes and Bell - indicates how valuable a revision of the other sheets of the Sydney field (sheets covering an area where mining has been intensive for half a century) would prove to be.

As a preliminary to the approaching development of the submarine extension of the Sydney coalfield, now proved, as this paper has attempted to show, over a frontage of 30 miles with unbroken continuity, there is required a mapping which shall show fully all undersea conditions proved by actual mining, including level contours, ocean-soundings, cover depths, and correlation of coal-seams, with structural sections: applying to elucidation of the structure of the field the new knowledge gained not only from actual mining, but from geological science in its present stage of progress. This is a work calling for co-operative effort of the mining companies, the government of Nova Scotia, the Geological Survey, and our own Society. These remarks are made not in any carping spirit, but as a plain statement of conditions calling for constructive effort. To avoid misunderstanding, it should be realized that the extent of the work required in the Sydney coalfield alone (which includes a new topographical base-map) calls for considerable expenditures, and possibly five years field and office work, plus a certain amount of prospecting by drills and otherwise.

The geological sheets covering the coal-basins in Inverness country are similarly incomplete.

THE FUTURE OF UNDERSEA MINING OF COAL IN THE SYDNEY FIELD

The persistently regular and comparatively gentle undersea dip of the coal seams in the Sydney field over a proved shorefrontage of 30 miles, and the general absence of disturbance of the strata, are conditions favouring undersea mining to an extent and over a period of time that will probably exceed all similar coal-mining operations elsewhere.

To what distance beyond the shore-line will it be found possible to mine coal in this submarine field?

The value of coal is relative. The economic limitation of production cost will doubtless, as previously suggested, operate in this field before physical limitations, considered merely as such, but it is not possible to forecast with any hope of accuracy how the market value of coal may extend or contract this limitation of production cost (1).

It can only be suggested that coal is being consumed at an increasing rate from a diminishing reserve, and that present indications point to a much wider range of uses for coal. Due regard is paid in this suggestion to the more efficient use of coal now being achieved.

As elsewhere noted, the seaward extension of existing collieries is limited by the inadequacy of the shore openings. A distance of three miles from shore has been used by operating companies in calculations of the workable reserve of coal tributary to existing collieries. The late Dr. Dowling in the 'Coal Resources of Canada' (²) included under the heading of 'probable reserves' the undersea coal within a three-mile limit from shore to a depth of 4,000 feet, and added also as 'probable reserves' undersea coal lying between three miles and five miles from shore at depths between 4,000 feet and 6,000 feet. It should be noted that this estimate is exactly as stated, being on the same lines as the calculation of reserves in other countries. It is not an estimate of *recoverable* coal.

⁽¹⁾ Compare "British Coal Trade", by H. Stanley Jevons; 1915, page 77.
(2) "Coal Resources of Canada", by D. B. Dowling; Geol. Survey of Canada, Memoir 59, 1915, pp. 9 and 10. Dr. Dowling's estimates were based on the work of a Canadian Committee which assisted in compilation of the estimates of coal reserves made by the International Geological Congress of 1913, q.v.

From experience so far gained in this field, it seems probable that depth of cover and consequent roof-pressure will prove a factor more limiting than distance from shore. The engineers of the operating companies have adopted 2,400 feet of vertical strata-cover as the limit of undersea extraction for purposes of reserve-tonnage calculations, but it is believed this limit will be exceeded. A depth of 2,300 feet has, as elsewhere noted, been attained in undersea mining at Inverness colliery; and at Springhill Mines, in a land area, a depth of 3,000 feet has been reached. In both instances, however, great difficulties from roof pressure are being met with.

The cost of production in the remoter undersea areas in this field will be a component of roof pressure and distance from shore, but roof pressure probably presents the greater difficulties.

It is suggested that the necessity of new methods of approach in the winning of remote undersea coal will arise with some urgency when workings have reached a distance of three miles from shore.

The suggestion was made twenty years ago by the late Geo. Blake, Walker that submarine coal in the Sydney field should be won by a deep shaft and a cross-measure tunnel intersecting the coal-seams in ascending order. At that date the submarine field, except the portion won from the Princess shaft at Sydney Mines, had barely been entered by workings, and the large capital expenditure required deterred the undertaking. The writer believes that earlier use of this method would, by resulting economies, have more than justified the expenditure.

At this date, with the submarine area proved by actual workings, the writer ventures to suggest this method for winning of the submarine coal lying beyond the economical extraction-limit of existing submarine mines and their projections, and for any new winnings situated upon shore frontages not now allotted to working collieries.

Figure 12 is a sketch of a cross-measure level tunnel, or tunnels, projected seawards along the axis of the Glace Bay syncline from the bottom of an assumed shaft 2,500 feet deep situated on Table head in the vicinity of the present No. 2 and 9 shaft of the Dominion Coal Company. The sketch is intended to illustrate the extreme application of the proposed method to presumed conditions in this district. The method is, of course, open to any desired modification as to depth of shaft; and as to length of tunnels, which would be determined by the grade selected, and the number of seams intersected. Probably, in practice, a slightly-rising grade seawards would be adopted.

To develop with even partial accuracy the feasible seaward extension of such cross-measure tunnels would require detailed studies and research entirely outside the scope of this paper, but it may be noted that in no instance of submarine coalmining that the writer has been able to learn of, are natural conditions so favourable to maximum use of intersecting tunnels as in the Sydney field.

The heavy expenditure that would be entailed in even a partial application of this method would not at this date be justified unless it resulted in winning a virgin tract, unconnected with the existing workings, and secured against an inbreak of the sea occurring at or about the shore-line. Because of the existence of shore workings of old date, which are uncertainly mapped, and because of active coast-erosion, elsewhere noted, prudent mining would suggest leaving a wide barrier of coal between the most seaward extension of existing collieries and the area designed to be won by cross-measure tunnelling, so as to secure a complete separation between the collieries having workings partly under the land area and partly under the sea, and the distant submarine area, the winning of which is now suggested as a future possibility (1).

As coal-seams contained within the strata section 'blocked out' by the level-tunnels projected in Figure 5 will all be within workable depth, the economic accessibility of the coal will depend upon large-scale rapid transportation of the mined coal from the working faces to the surface. Given a sufficiently large cross-section of tunnel, the employment of electric trolley-haulage, with large capacity cars and double-track laid with heavy rails and ballasted, the rapid movement of coal

⁽¹⁾ See reference to barrier left against flooded area in Chilean instance.

should present no great difficulty. Stone-dusting and the use of the haulage tunnels for intake air should provide reasonably safe conditions in a stone drift.

The entire extent of the tunnels, and the workings to which they will give sole entrance and exit, will form almost a closed circuit, having a point of admission and exhaust to fresh air separated only by the distance between the downcast and upcast shafts.

The tunnels will have to ventilate mine workings of increasingly large area in several seams, and while the maximum circulation of air will not be required in the earlier stages of winning seams in the outer areas, the airways will require to be driven large enough in the first instance to supply the eventual maximum volume of air.

The areal extent of the tunnels will therefore be decided primarily by the factor of ventilation, to which the arrangement of separate ways for haulage of coal and of men, and for transmission of power and conveyance of water, will be subordinated.

It is suggested that the shafts and air-passages will require to be of sufficient area to circulate up to one million cubic feet of air per minute, and that, to obviate impracticable watergauges, the provision of a forcing fan at the downcast shaft, of a booster fan 'inbye', and of an exhaust fan at the upcast shaft, will all be necessary eventually.

While a cross-measure tunnel of such magnitude does not appear to have been used in coal-mining, technical literature records tunnels in metal-mining for transportation and drainage, of lengths up to 24,000 feet, in Colorado, Utah, and Japan (¹). It is clear, however, that the problems of ventilation connected with these tunnels are not comparable with those to be worked out in tunnels such as are herein suggested, the circumstances of which are unique.

Whether the driving of separate tunnels will be found advisable, or whether one tunnel of sufficient area to permit of partitioning-off into separate passages will be admissible, can be developed only by detailed studies.

⁽¹⁾ See Brunton and Davis, "Modern Tunnelling", 1914, pp. 35-52; also "Ore and Stone Mining", by Le Neve Foster and Cox, pp. 462-464.

The frontage of the 'intake' allotted to a cross-measure winning of so costly a character would require to be wide, possibly not less than three miles on either side of the tunnels, hence extensive auxiliary level-haulage would be required. The question of motive power for such haulages and for coalcutting is one that the possibility of remote undersea mining will largely depend upon. For all purposes except use at the coal-face, electricity appears to be the only possible power in the circumstances (1). Electrically-driven air-compressors, suitably housed, and ventilated by a split of fresh air from the intake, giving a positive outward pressure, could be used to provide power for coal-cutting and face-haulages, if provision for circulating-water for cooling should be found possible. (See reference to Whitehaven colliery, page 95).

This admittedly crude sketch of a future undersea winning, of the unusual nature herein suggested, is attempted because of its bearing upon existing operations and immediate projections, and also because it will serve to illustrate the large expenditures that will be required if the unmined reserves of undersea coal are to be fully utilized.

It should be fairly evident that an undertaking of the suggested magnitude will not be possible under existing lease-tenures or royalties. Indeed, it may very well happen that the Government of Nova Scotia will find it advisable, in order to maintain the coal-mining industry, to assist in the expenditures required by such an enterprise, and to greatly modify or entirely forego royalty collections on coal mined under such conditions.

Furthermore, the existing method of leasing of submarine areas, clearly foreseen fifty years ago to be inapplicable to mining of coal under the sea, is still less applicable to the remote undersea reserves.

The right to grant, by leases, access to undersea coal seams, is derived from possession of the foreshore (2), but it is evident that the collection of royalties or wayleaves on coal

⁽¹)''Coalfields and Coal Industry of Eastern Canada''; Mines Branch Dept. of Mines, Bull. No. 14, page 43.
(²) See Caleb Pamely, "Colliery Manager's Handbook".

mined from areas far out under the sea will have to be modified in relation to the cost of mining and removing coal through the foreshore openings.

Such an undertaking as is adumbrated in the foregoing will necessitate close co-operation between the Province of Nova Scotia and those who seek to operate the remoter submarine coal-areas, and unquestioned access to all seaward coal so far as leasing agreements are concerned.

The parallelism between the forecast of future undersea mining in the Sydney Field, herein attempted, and the remarks of Mr. J. S. Dixon on the future of mining under the Firth of Forth, will be noted (1).

While the present forecast has been independently developed, this parallelism is the result of similar physical and economic factors, but it should be pointed out that in Cape Breton, in contra-distinction to conditions in Scotland, the submarine reserve of coal is already more valuable than the land areas, and accessible with comparative ease.

THE WEST SHORE OF CAPE BRETON ISLAND

On the western shore of Cape Breton island, within a coast frontage of 80 miles, there occur four, detached, coal-basins. The presence of *anthracomya* shells suggest that the seams are of the same general age as the upper seams in the Sydney coalfield. No attempt at correlation of the seams in these separated basins has been made. The land exposures are in each instance limited, almost fragmentary in fact, the major portion of the coal deposits occurring under the sea.

While it was formerly supposed, possibly by analogy with the Sydney coalfield, that these detached basins were vestigial land-exposures of a large coal-field hidden beneath the waters of the gulf of St. Lawrence, little evidence of this exists, and there seems rather more evidence in favour of the origin of these coal occurrences in unconnected and local areas of deposition.

Port Hood:

This almost entirely submarine trough of coal measures—the most southerly of the four basins mentioned (See Figure 1)—has a shore frontage of two miles, with one workable

⁽¹⁾ See page 100.

seam outcropping a few hundred feet above high-water mark. This seam, from 6 feet to 7 feet in thickness, dips under the sea at an inclination of 21 degrees, flattening to 12 degrees in the lower workings.

The outcropping of a higher seam, doubtfully said to be 6 feet thick, is mentioned by Richard Brown as visible at low tide in 1871, but the existence of such a seam was not proved from the workings in the lower seam.

Workings on pillar-and-room method had proceeded under the sea for about 2,500 feet on the full dip when, in 1911, the mine was flooded with salt water and has not been recovered. Water made its appearance in the roof at a point where a pillar was being drawn in the lowest level of the mine, just inbye of the slope pillar, with a solid cover of 942 feet. The leak was at first small, but rapidly increased, being estimated at a flow of 3,000 gallons per minute in the initial stages. As this flooding followed an inbreak of the sea in 1909 in the neighbouring Mabou district (elsewhere referred to), an investigation was made at the instance of the Commissioner of Mines (1). The Commissioners found a mean difference of 0.82 ft. between high and low water-mark in the mine, coinciding exactly in time with the occurrence of high and low water-mark at the shore, where the mean difference of tide-levels was 3.14 feet. At a later date, the water-level in the mine remained constant. The report found that there had been no violation of good mining practice in the extraction of the coal, but recommended that "in future every reasonable means should be employed to ascertain the depth, nature and condition of the overlying strata before pillars are extracted in any submarine area." The conclusion of the Commissioners was:

"that the mine is flooded by the water of the ocean, and we are more inclined to believe that it was through a fissure in the rock more nearly vertical in direction, than that the water followed the strata a long distance, finally making its appearance in the mine at the point above described."

⁽¹⁾ Report of Inspector of Mines and Appraisers, Port Hood Mine, Mabou Mine, 1913; King's Printer, Halifax, N.S.

Rather extensive pillar-drawing had taken place in a higher level in the mine at a date much earlier than the inbreak. In regard to reported movements of the strata consequent upon this extraction of pillars, the Commissioners stated:

"We are by no means convinced that the movement, if any, ever was pronounced enough to be called a 'creep', and we are not of the opinion that this so-called creep in any way contributed to the flooding of the mine".

With regard to the actual cause of the flooding, the conclusions of the Report are negative, and possibly no satisfying explanation of the break in the roof is now possible.

The suggestion of the writer, made at the time of the investigation, is that the draw of the roof above the extracted pillar-area in the third level so opened up the overlying measures as to cause an accumulation of water in the strata, fed by gradual percolation of sea-water down the jointing of the metals, and gaining original admission where the strata planes crop out in the sea-bottom. The drawing of a pillar at what was virtually the deepest place in the mine, and therefore exposed to the greatest head of pressure from any water contained in the strata, occasioned, it is surmised, a breakthrough of a large volume of salt-water indirectly connected with the sea — probably between high and low-water marks — and presumably controllable with adequate pumps.

The writer is unable to agree with the conclusion of the Report regarding a vertical break of the strata, because this involves admission that extraction of a single pillar (1) in a seam under seven feet thick, could, without great lapse of time, cause a vertical crack of the strata extending 942 feet to the sea-bottom, whereas much more extended pillar-drawing, of much earlier date, at a point nearer the sea-bottom, was not followed by any inflow of water into the coal-seam itself.

This unfortunate occurence caused a total financial loss to the proprietors of the mine, and no attempt has been made to ascertain whether the leak is a controllable one. There is reason to suppose that with suitable pumps this may be the case. The Commissioners advised that unwatering should not be attempted, believing that in time the fissure through which the water made its way may silt up.

⁽¹⁾ See under Discussion, p. 128.

An unworked area of the seam lies to the south of the flooded mine, so situated as to permit entrance into the remaining undersea tract by a new winning.

Mabou Area:

At Mabou, ten miles north of Port Hood, pre-Cambrian hills form a semi-circle around fragmentary exposures of the rim, less than a half-mile across, of a submerged synclinal-trough of coal measures dipping at an inclination of 40 degrees along the axis of the syncline. At 500 feet from the outcrop the dip flattens to 20 degrees. The basin is delimited on each side by a fold or fault exposing the underlying gypsum. The inclination of the seam increases towards the fault, being almost vertical on the south side and 60 degrees on the north side. As the faults diverge seawards, it is thought that the basin widens under the sea. The measures containing the coal seams are laid down on lower Carboniferous rocks, which are metamorphosed where they contact with the pre-Cambrian rocks on the slope of the hills first mentioned.

Three seams are definitely known, respectively 6 ft. 6 in., 7 ft., and 12 ft., in thickness, of which the first two have been worked. Three other seams 5 ft., 4 ft., and 3 ft. thick are reported, but have not been actually proved.

The first-named (6 ft. 6 in.) seam was mined by slope workings for 2,000 feet down from the outcrop, where this seam and the 7-foot seam below came almost together, and later workings were chiefly on the lower seam because of its better quality.

The slope referred to was driven with only 110 feet of cover below the sea-bottom. Through a headway carelessly driven up the pitch towards the submarine crop, the mine was flooded in January, 1909, and has remained so. It is to be noted that the break did not occur in the slope.

The Commissioners who investigated the flooding came to the conclusion that it was an error of judgment to have entered the seam under such light cover, advised against attempt to unwater the slope, and recommended that any future workings should be opened on one of the lower seams.

Recent plans for operation of this basin, which did not reach fulfillment because of financial difficulties, contemplated the sinking of a shaft and drifting across the measures, thereby tapping all the submarine seams under safe cover.

This coalfield has been greatly depreciated by the unskilful method of attack first used. The later plan mentioned is one that would permit of a maximum yield of coal over a long period at moderate cost, and is one that competent engineers would adopt after careful preliminary proving of the seams, but the initial expenditure would be large.

The coal in the seven-foot seam first mentioned is reported to contain only 2 per cent sulphur and 4 per cent ash, and is in this respect a remarkable exception to all other seams on the west coast of Cape Breton, not excepting the other seams in the Mabou basin itself.

The Mabou basin, containing over 30 feet of coal in 550 feet of strata, has undoubtedly much potential value, but its full utilization will probably have to await conditions that will attract the very considerable capital investment necessary to develop this unique deposit in a workmanlike and approved manner. The flooding of the most accessible portion of the basin has made it impossible to prospect the seams under the sea at reasonable cost, a necessary preliminary to large expenditures on a shaft and cross-cutting. The field is of a type that has its value in the future, and prudent management from the lessor's viewpoint suggests its reservation intact, as far as that is now possible.

Inverness Coalfield, Cape Breton Island:

This field, known also as the Broad Cove field, is a narrow syncline of the Productive Coal-Measures lying upon pre-Cambrian rocks at the southern end, and upon lower Carboniferous rocks on the north. The seams have a land outcrop 5.4 miles wide at the shore, and extend inland $1\frac{1}{2}$ miles.

The seams dip due north under the sea at an angle of 20 degrees, the inclination increasing seawards. On the east, the field is cut off by an up-thrust of the underlying gypsum, the seams lying vertically at the fault-line. On the west, the inclination of the seams increases under the sea, reaching 70

degrees at the face of the workings in this direction. Twelve seams are reported to exist, but only four seams reach 4 feet and over in thickness, and two seams only have so far been worked.

A partial section of the measures is as follows:

Strata with three thin coal seams exposed at the shore	
'Thirteen Foot' seam (being mined)	4 ft. 6 in.
Strata (including a seam 1 ft. 6 in. thick)	184 ft.
'Seven Foot' seam (extensively mined)	6 ft. 4 in.
Strata (including three thin seams)	238 ft.
Seam (not mined)	4 ft. 0 in.
Strata (including 2 ft. 8 in. seam)	460 ft.
Port Ban seam (not mined)	5 ft. 0 in.

The 'Seven Foot' seam maintains a fairly regular thickness of 6 ft. 4 in., and the so-called 'Thirteen Foot' seam was proved by a rising drift from the Seven Foot seam below.

No. 1 colliery of the Inverness Coal & Railway Company, working the Seven Foot seam, has been in operation twenty-six years. The seam is entered through a slope opening, the mouth of which is 1,500 feet from shore. The face of the slope was, in 1926, distant 5,350 feet seawards from shore. The slope enters the submarine area with a cover of 240 feet, dipping at 20 degrees and becoming steeper, the inclination having increased to 55 degrees at the bottom of the slope.

The sea bottom has an average inclination of three degrees. The maximum solid cover under the sea so far achieved is 2,300 feet (1).

The seam has been mined entirely by hand-mined pillarand-room methods. Pillars are being recovered under the sea with solid cover of 725 feet, and there has been no indication of breaks.

No water is made in the submarine area, but a good deal of water finds entrance into the mine from the land workings.

The combined conditions of depth of mining, angle of inclination of the seam, and distance of haulage uphill, are, the writer believes, without precedent in undersea mining, and are, of course, increasingly onerous. The main haulage-road traverses the seam at half a right angle to the true dip, and is inclined at 28 degrees at the bottom lift.

⁽¹⁾ Compare with Balmain Colliery, Sydney Harbour, N.S.W., page 105.

The upper seam now being worked was proved in 1920 by a cross-measure drift 700 feet in length driven upwards from the Seven Foot seam. At the shore-line this seam has a cover of 430 feet, but because of the relation of the coast-line to the inclination of the seams, and their submarine outcroppings, it will be necessary to drive further seawards before gaining sufficient cover for extensive submarine winning. Very little has been done in this seam under the sea, except proving by the tunnel (1).

The composition of the strata containing the coal-seams is as follows:

	Per cent
Boulder clay and earth	3.0
Shales	24.7
Sandy shales	35.0
Shaly clays	6.0
Fireclay	5.0
Sandstone	
Coal	

Chimney Corner:

Twelve miles north of the Inverness mine is a ridge of high land extending 2½ miles along the shore, having an elevation of 300 feet above sea-level and containing the outcroppings of two, or, less certainly, three, coal seams, dipping at an angle of 35 degrees under the sea (2).

The geological structure and the character of the coalseams are not well determined, nor is the thickness of the strata cover, where the seams enter beneath the sea, precisely ascertained.

While a submarine reserve of recoverable coal of small extent is believed to exist, it is unlikely to be developed under present economic conditions.

South of Chimney Corner is the Ste. Rose coalfield which. until the structure was partially elucidated by the officers of the Geological Survey, appeared to be continuous with the Chimney Corner outcroppings, and to early observers suggested

⁽¹⁾ Note: For particulars regarding the submarine workings of the Inverness Railway & Coal Company, and the composition of the strata, the writer is indebted to Mr. D. Rorison, Manager.
(2) Geol. Survey of Canada, Summary Report 1918, part F, by A. O. Hayes; page 8. Description of the Chimney Corner and Ste. Rose coalfields,

with map.

a coalfield with a submarine extension extending perhaps along a shore frontage of five miles. It is now clear that the Ste. Rose area is an egg-shaped synclinal basin outcropping inland, and having no submarine extension. The basin is limited by a gypsum fault paralleling the shore, and is divided from the Chimney Corner seams by a wide and very disturbed fault-zone.

JOGGINS AREA, NOVA SCOTIA

An extension of the coalfield of Cumberland county has been worked under Chignecto bay, an arm of the bay of Fundy, in the Joggins district. The seams outcrop regularly over a distance of twenty miles along the northern edge of a syncline with its axis approximately at right angle to the shore.

At the shore occurs the section of coal and associated sediments made famous by Logan's Section of 1843, and by the writings of Dawson. Between seventy and eighty seams of coal, all comparatively thin, are exposed, showing a rich assembly of fossil trees and plants. In a distance of half a mile along this section, the Geological Survey of Canada record the presence of 73 erect trunks of *sigillaria* and *lepidodendron*



Figure 13.—Fossil tree and root exposed at low tide near Joggins, N.S.

trees, many of them rooted in situ (1). Figures 13 and 14 are reproduced from photographs kindly furnished by Dr. Chas. M. Sternberg of the Geological Survey (2).

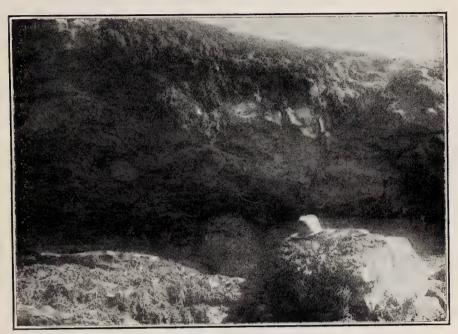


Figure 14.—Fossil sigillaria on the Joggins shore. Low tide, Bay of Fundy.

Out of the numerous seams only six are workable, and one only has been followed under the sea. This is the Joggins Main Seam, which has a top section 3 ft. 6 in. in thickness and a lower section 1 ft. 6 in. thick, separated by a band of clay of varying thickness ranging from six inches to 24 feet. Only the upper section has been worked under the sea, the clay band being too thick to permit of both sections being mined together.

The seam is inclined at 17 degrees. The sea floor is In the submarine area, pillar-and-room virtually level. extraction was employed because of thin cover. Further out from shore, advancing longwall was adopted, with electricallydriven mining machines, with entire success.

^{(1) &}quot;These standing logs have been admired by all geologists since Richard Brown discovered them in 1829, and the drawings of them by Logan, Lyell, and Dawson have been repeated in most text-books on Geology." Pirsson and Schubert, "Historical Geology", 1915.

(2) In the Victoria Museum at Ottawa is an assemblage of fossil trees, the position in which they was found assemblage and the control of the control

reproducing the position in which they were found associated with the coalseams at the Joggins shore.

Unfortunately, as the workings advanced seawards the seam became thinner, and was invaded by dirt partings, and it became evident that the termination of the field was being approached. The workings had advanced about 4,000 feet from shore when the operation was perforce abandoned. The cover at this point was 1,500 feet. No change in the inclination of the measures accompanied the failure of the coal-seam.

Other seams being worked inland two miles from shore become very thin at the shore-line, strengthening the indications of termination of the field under the bay of Fundy.

The Joggins submarine operation, while small in extent, is noteworthy in Nova Scotia because it proved the economic availability of a thin coal-seam extending under the sea, by employment of machine-cut longwall workings and complete electrification of all underground operations, including coalcutting. The operation was in this respect the most modern in Nova Scotia (the conditions being of course unusually suited to longwall extraction), and it is therefore matter for regret that the failure of the coal-seams has compelled abandonment of the submarine workings (1).

This instance is further evidence of the tendency of the coal deposits in Nova Scotia to occur in localised basins of deposition.

GOVERNMENT REGULATION OF UNDERSEA COAL-MINING IN NOVA SCOTIA

Method of Leasing Coal Areas:

The province of Nova Scotia has had complete ownership of its coal seams since 1857, and control of coal-mining has not only been exercised through ordinary legislative enactments, but has been possible through lease covenants. Notwithstanding the early warnings of competent advisors, such as Dr. Poole, the legislatures do not seem to have foreseen the future importance of the undersea coal, and no distinction was made in the laws governing leasing as between land and sea areas. On land, the leasing of coal areas having arbitrary boundaries, at a set rental per acre or per square mile, is

⁽¹⁾ The foregoing particulars of the Joggins submarine winning have been kindly furnished by Mr. N. T. Avard, General Manager of the Maritime Coal & Rly. Company, Joggins, N.S.

customary, but the application of this method to undersea areas is illogical, and has been wellnigh disastrous, because it ignores the factor of accessibility at the shore, one of prime importance in regard to the potential value of coal under the sea. In this regard, the legislatures appear to have lost sight of the necessity and the inherent right of a lessor to prescribe such covenants in leasing as shall ensure the easiest access to the leased property and the maximum royalty-revenue — or, in other words, the maximum tonnage-yield from the property over the maximum period. The undersea areas should have been plotted off along the shore line into mining areas having adequate seaward extension, with the land continuation of the undersea lease sufficiently included to ensure full accessibility, and room for the most convenient openings into the seams.

Instead, leases were given of shape parallel to the shore, further out to sea than, and blocking the seaward progress of, workings in leases of earlier date. In one notable instance the lease line followed the shore-line for many miles, separating ownership of mining rights to land and sea coal.

It seems evident that the leasing practice, as did the mining practice, followed unthinkingly from land usages to novel conditions under the sea, without adaptation thereto. Not that the future was unforeseen, because, to quote Dr. Poole once more, he wrote in 1877:

"For land areas the system [of leasing] adopted answers well enough perhaps, but now that the general course of the coal beds is approximately known, and the future value of those under the sea recognized, it is most apparent that the system is not best suited for the submarine. Instead of allowing distinct individuals to take out leases of areas, one beyond the other, it would undoubtedly have been better, and more conducive to the interests of the country, to have restricted each lessee to a certain frontage on the adjoining coast, taking into consideration the outcrop of the seams rather than a given superficial extent."

The formation of the Dominion Coal Company, in 1893, consolidated the leases of connected land and sea areas on the south side of Sydney harbour, those on the north side remaining in possession of the General Mining Association until 1900, when the Association sold its remaining holdings to the Nova Scotia Steel & Coal Company. In 1906, litigation between the Dominion Coal Company and the Dominion Iron & Steel

Company occasioned activity by the Steel Company to secure coal areas, and a general desire to acquire areas and suitable sites for mine openings manifested itself, which ended by the three companies named becoming lessees of all the undersea coal.

About 1913, the workings of the Princess colliery had reached the lease boundary ("the extreme limit to the dip mentioned by Mr. R. H. Brown") and an arrangement was come to between the Nova Scotia Steel & Coal Company and the Dominion Coal Company by which a strip of coal one mile wide by two miles long was sub-let to the Scotia company, giving this company a corridor to a block of leases, the nearest point of which lay four miles from shore.

In 1918, the Scotia company found itself with so small a tonnage of ungot coal remaining inside the boundaries of the original General Mining Association's undersea lease as to necessitate the speedy closing of the Princess and Florence collieries unless the boundaries could be enlarged. The difficulty of mining in the sub-let area in the Princess colliery, occasioned by weight of the strata and the haulage problem, was such that the main deep made only slow advance towards the outlying areas.

The lease ownership of the undersea coal had meanwhile been reduced to two parties, because of the consolidation of the Dominion Coal Company and the Dominion Iron & Steel Company into the Dominion Steel Corporation.

A suggestion to exchange lease areas so as to give the Dominion companies possession of all the undersea leases on the south side of a line drawn through the centre of the entrance to Sydney harbour, and to the Scotia company possession of the undersea leases on the north side, was found impossible of arrangement at the time.

The Legislature of Nova Scotia, recognizing that the difficulty arose out of the method of leasing, and that the Province, as lessor of the undersea coal, could not, except to its detriment, permit the indefinite interference with the seaward advance of mine workings which might result from the inability of lessees to come to arrangements permitting such advance, passed an Act, the summarised provisions of which are as follows:

The Provincial Government may appoint two or more commissioners to enquire whether any submarine area held under lease can "if unworked be advantageously worked in the best interests of the Province by some other lessee", and is given power to transfer areas to ensure that undersea leases "shall be worked in the best interests of the Province", proper compensation being prescribed. The Act does not apply to the 99 years 'blanket leases' held by virtue of special legislation incorporating the Dominion Coal Company (1), and dating their term of tenure from 1893.

In 1920, the incorporation of the British Empire Steel Corporation brought the whole of the undersea areas under single management. The Dominion Coal Company had, in 1913, acquired the areas of the North Atlantic Company in the Morien basin, so that it became possible for the first time to view the engineering aspect of the submarine field as an entity, and to recast mine projections without regard to lease boundaries.

It is important to note, also, that a formal exchange of areas between the companies interested has been arranged, carrying out, with some substantial improvements, the suggestion referred to of a division on either side of a line dividing Sydney harbour, so that an alteration of corporate structure will not create the crippling situation that previously existed.

It is matter for congratulation that the difficult situation into which the Province as lessor of the undersea coal was placed by the lease system has been to this extent ameliorated.

The immediate result of the removal of lease-boundary hindrances through unified management was the development of Dominion No. 1B colliery, which has allocated to it a territory including three lease-interests, and has as a future field of operations an indefinite seaward reserve of coal.

A far more important result is that the way is now clear, in mining of the undersea coal, for engineering plans of that broader and novel character that will follow, now that the tentative and experimental stage of submarine mining has

^{(1) &}quot;An Act respecting submarine areas", May 17th, 1919. Nova Scotia Statutes.

passed — plans that could never be perfected had the lease boundaries been allowed to remain and interfere with the work of the engineer.

GOVERNMENT REGULATION OF MINING PRACTICE

The laws which have prescribed the method of mining in undersea areas in Nova Scotia have been marked by more prescience than the haphazard method followed in leasing the areas, and there has always been held in view the necessity for "so conducting all inshore mining that ultimate deep-sea mining may be safely prosecuted" (¹).

Up to the time that the General Mining Association commenced undersea mining in Nova Scotia, there had not, so far as the writer can ascertain, been extraction of coal from beneath the sea anywhere except in Great Britain. The regulations governing undersea mining in Nova Scotia were drafted by Dr. Poole in 1877, and were based on British experience and largely upon the evidence bearing upon submarine coal-mining given before a British Royal Commission on Coal Supplies. The clauses in the Coal Mines Regulation Act remained unaltered until 1924, when the section was amended, as shown in detail in parallel columns in Appendix 'A'.

It had always been admitted that the submarine clauses in the Coal Mines Regulation Act were tentative, and that hard and fast statutory regulations could not be made while so much remained to be learned from experience. The progress of the mine workings to sea, and the opening of several new mines, with almost entirely submarine territory, about 1907, raised questions which caused the Nova Scotia Government to engage Mr. T. E. Forster, of Newcastle-on-Tyne, for the purpose of reporting on the suitability of the regulations to the conditions under which undersea coal was then being worked in the Province.

⁽¹⁾ H. S. Poole, Mines Report, 1877; Halifax, N.S.

It is significant that Mr. Forster in his Report (¹) prefaces his technical recommendations by reference to the method of leasing in Nova Scotia, which he compared to regulations in other countries. Referring to the British practice he remarked:

"There are no clauses in the Coal Mines Regulation Act (British) dealing specially with submarine areas, as in Nova Scotia; any special provisions as to working being included in the several leases. In framing these conditions it has always been the practice, and I think a sound one, to consider the special circumstances of each letting, rather than an attempt to regulate the whole on a hard and fast basis".

Mr. Forster recommended the elimination of the clause prescribing a panel system of undersea mining, pointing out that, beyond the 500-feet cover-line, the system of working is unfettered, though subject to the control of the Inspector of Mines; and he suggested that (except for the provision that no coal shall be wrought with a less strata cover than 180 feet), the system of working should be entirely under the control of the Inspector of Mines.

Further recommendations were that in longwall operations an exploring drift should be driven fifty yards at least in advance. Also, that surveys and levellings of undersea workings should be made every three months, and that the levels and depth of cover should be marked on the mine plan, together with records of soundings; regulations which confirmed the existing practice of the Dominion Coal Company.

Mr. Forster again emphasised the point made by Dr. Poole thirty years previous by stating:

"I am strongly of the opinion that the conduct of undersea workings cannot, generally speaking, be advantageously provided for by hard and fast rules, and that a careful consideration of the circumstances of each case, guided by experience gained in the gradual development of operations, is the proper system to be pursued in such cases."

The amended regulations of 1924, it will be seen, follow Mr. Forster's recommendations very closely, but in the sixteen years that elapsed between the date of his Report and the enactment of new regulations, and since, the mining practice has been governed by the full discretionary powers given to the Minister of Mines, exercised in consultation with the mining engineers and officials having direct oversight of the working of the undersea collieries.

⁽¹⁾ T. E. Forster, "Report on Submarine Coal Areas", 22nd October, 1908; Nova Scotia Mines Report, 1908.

SUBMARINE COAL-MINING ELSEWHERE THAN IN NOVA SCOTIA

British Columbia

Nanaimo Area:

An area of coal-bearing rocks of Cretaceous age occurs along the eastern side of Vancouver island. The area is separated by an exposure of crystalline rocks, 12 miles in width, into a southern portion, having its most important economic development in the Nanaimo district, and a northern portion in the district of Comox. The Nanaimo area has a coast frontage of 40 miles, the measures, with included coal-seams, dipping generally seawards.

Mining was first commenced in British Columbia by the Hudson Bay Company at Nanaimo in 1852. The Western Fuel Company, in 1902, acquired the coal interests from the Hudson Bay Company and has in the meantime conducted mining operations under a large part of Nanaimo harbour (See Figure 15).

At this date — 1926 — about 20 per cent of the coal production of Vancouver island, or over 320,000 tons per year, comes from undersea workings in the Nanaimo district.

Two seams are worked, the Douglas seam, and the Newcastle seam lying from 70 to 100 feet below. Both the seams and the associated sandstone and conglomerate strata are irregular in thickness.

The No. 1 shaft of the Western Fuel Company, situated near the shore at Nanaimo, reaches the Douglas seam at a depth of 640 feet and the Newcastle seam at 700 feet. The extended workings of this colliery are reached by another shaft on Protection island. The workings in the upper seam, which has been mined chiefly by pillar-and-room workings, are extensive, being four miles in width and reaching seawards up to two miles from shore. The pillars in the upper seam have been partially extracted, and there are a number of barren areas several acres in extent. All workable portions of the Douglas seam are fully extracted except where the overhead cover becomes less than from 375 to 400 feet in thickness below the sea-bottom. The further seaward advance of the Douglas

seam workings was halted some ten years ago by deterioration of the seam, which runs out into black shale, as indicated in Figure 15. At the present time, operations in the Douglas seam are directed to recovery of coal in areas passed by in the

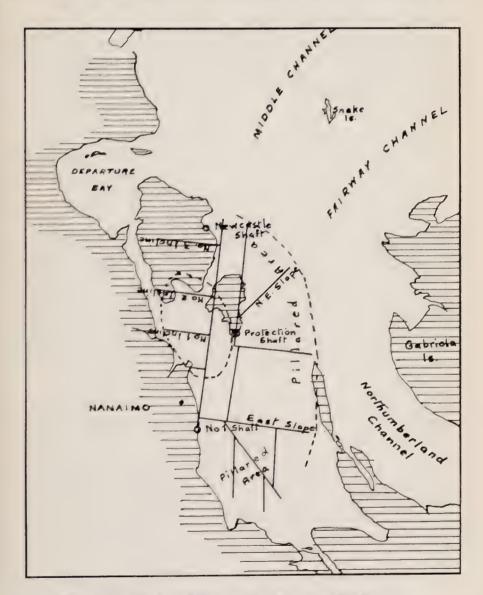


Figure 15.—Coal workings under Nanaimo Harbour, Vancouver Island.

Outer dotted line shows where advance of workings stopped because of thinning and deterioration of the Upper, or Douglas, Seam.

The egg-shaped dotted area nearer shore shows approximately the longwall workings in the underlying Newcastle Seam.

first working as unfavourable, but which more intensive prospecting has proved to be workable. Some thin areas of the Douglas seam are being now worked by longwall methods.

The lower, or Newcastle seam, is mined exclusively by longwall method. The thickness of the coal varies from 2 feet

to 6 feet, and averages 3 feet.

The sea-floor is covered with mud and clay, which acts as a seal. The harbour is shallow and free from strong currents. Small feeders of water have been experienced at times in pillar areas, the heaviest being a leak of about 75 gallons per minute that ceased after running six months.

A number of folds cross the field, without actual dislocation of strata, and small faults are encountered. The seams have been mined with covers of from 400 to 1,500 feet. No undue

difficulties have been met with from roof pressure.

Electricity is used for haulage, electric locomotives of overhead trolley-wire type being used. Electric power is sparingly used off the main-haulage levels, and compressed air is used for coal-cutting and other purposes at the coal-face.

Ventilation problems have been aided by the presence of islands, the Nanaimo undersea field being in this regard uniquely favoured. An exhaust Sirocco fan at No. 1 shaft on Vancouver island delivers 150,000 cu. ft. per minute against a 5-inch water gauge, and another fan at the shaft on Protection island delivers 75,000 cu. ft. at 1.75 inches of water gauge.

The three shafts and their tributary workings are all connected. The travel of men to work is minimized by use of the Protection island shaft as a man-shaft.

Undersea coal-mining is regulated by statute. The text of the provisions in the Coal Mines Regulation Act is given in Appendix 'B', and it will be noted these are similar to the provisions in the Nova Scotia Act. An additional provision in the British Columbia Act mentions possible winning of undersea areas by "caisson, shaft or concrete revetment".

There are areas north and south of Nanaimo, approaching the rim of the Cretaceous trough, where the solid cover over coal-seams is less than that required by the statute, and in one instance, ten miles north of Nanaimo, further general mining was prohibited by the Inspector of Mines because of insufficient cover. The British Columbia laws allow purchase of coal areas, but a royalty is paid to the Province on coal removed for sale. The Western Fuel Company's areas at Nanaimo are held in fee simple.

Suquash District:

An area of coal-bearing sandstones and shales, 164 sq. miles in extent, occurs on the west coast near the northern end of Vancouver island. The seams are contained in a broad syncline with gentle dips seawards ranging from level to ten degrees. The age of the deposit is believed to be slightly younger than the Nanaimo series, and the coal-seams are more regular than at Nanaimo and Comox. There are a number of coal-seams present outcropping along the shore, of which two only are known to be thick enough to be workable.

A shaft sunk by the Pacific Coast Coal Mines, Ltd., was located 150 feet back from the shore and proved at a depth of 160 feet a seam which was $6\frac{1}{2}$ feet thick where worked by longwall method. Operations have been suspended at this property, for financial reasons, since the outbreak of war in 1914, but at that date a longwall face 1,500 feet in length had been developed. A seam 4 feet thick was indicated by boring at 282 feet below the seam worked. The upper seam showed bands of 'bone' coal and shale, which lessened in extent going to the dip.

The inclination of the seams is less than one degree over the immediate shore-frontage developed by the workings mentioned, giving a solid cover below the sea-floor of only 130 feet over a large area of the upper seam. The lower seam, if economically workable, would have sufficient cover, but the operation did not proceed beyond the initial prospecting stage, and did not enter the undersea area.

The Suquash coal is non-coking and comparatively smokeless, making an ideal domestic fuel.

There is a probability of a future submarine mining operation of some importance in this district (1).

⁽¹⁾ Note: Much of the foregoing particulars were supplied by the late Mr. George Wilkinson, formerly Chief Inspector of Mines in British Columbia, and have been supplemented by information kindly furnished by Mr. J. D. Galloway, the Provincial Mineralogist, and the Chief Inspector, Mr. Dixon.

GREAT BRITAIN

As the early history of coal-mining was chiefly made in Great Britain, it is in that country that the earliest instances of undersea mining occurred, and in other countries, where coal has been mined under the sea, British engineers have either been responsible for commencement of the undertaking, or have largely influenced the practice.

Cumberland Coalfield:

This coalfield extends along the southern shore of Solway firth for 15 miles, approximately from Maryport to St. Bee's head.

About one million tons of coal annually is taken from submarine areas in this field. The Whitehaven collieries produce 600,000 tons, and the Harrington collieries 200,000 tons, per year, from workings extended far out under the sea, and there is a considerable output from submarine workings at the St. Helen's and Risehow collieries, which are not so far out to sea as the Whitehaven collieries.

The Whitehaven collieries are in many respects the most interesting example of undersea coal-mining practice. Coal has been worked under the sea at Whitehaven since the 17th century, and it is believed was the first instance of the practice. At Whitehaven, also, workings have been extended further under the sea than elsewhere.

It was in the neighbourhood of Whitehaven, at Flimby, close to Workington, in 1834 that the sea broke into the workings of the Lady and Isabella pits and drowned 36 men and boys.

These collieries worked a ten-foot seam at a depth of 600 feet for a distance of 4,500 feet under the sea. Workings had been driven to the rise until the distance to the sea-floor overhead was 120 feet. The rooms were driven 15 feet wide, leaving pillars from 21 ft. to 24 ft. wide. These inadequate pillars were robbed, with a result that is said not to have been unexpected. The depth of water over the break in the sea-floor was 30 feet, with a depression of 60 feet within the fracture, stated to have measured 90 ft. by 240 ft. in area (1).

⁽¹⁾ Dunn's "Treatise on the Winning and Working of Collieries, 1852".

The general dip of the strata in the Whitehaven district is seawards. A system of faults, some upthrows and some downthrows, crosses the measures. In a distance of slightly under four miles, proceeding from shore, the sum of the several downthrows is 438 feet, which, is offset by upthrows totalling 330 feet. The effect of the faults so far met with in the seaward advance of the workings has been to increase by over 100 feet the gain in depth of cover resulting from the inclination of the seams. The depth of the sea is fairly constant, being from 80 to 100 feet over the sea-area likely to be coal-bearing, a depth which is maintained at $3\frac{1}{2}$ miles from shore. The sea-floor, which is very flat, consists of sticky clay and sand, forming an excellent seal over the underlying rocks. The measures contain a number of underclays which also act as seals.

At Whitehaven the seams dip seawards about 4 degrees. Workings have extended four miles under the sea from highwater mark, depth of cover being at this point 1,400 feet with 100 feet of water. (See section of workings, Figure 16).

Both longwall and pillar-and-room extraction are employed. Generally speaking, longwall is practised in seams up to 5 feet in thickness, and pillar-and-room in seams exceeding this thickness.

The thinnest seam worked at Whitehaven is 2 ft. 3 in. in thickness, and the thickest 15 feet.

Longwall extraction has been tried in the thicker seams, but proved impracticable on account of the heavy rock-roof. At the present time, ninety-five per cent recovery of the coal is attained. In room-and-pillar workings, pillar extraction follows quickly upon the room-work, the 'whole' and the 'broken' being really worked together.

Four superimposed seams have been worked under the sea. The order of extraction has at Whitehaven, as elsewhere, depended upon the quality of the seams, the best seam being first worked. At no time have superimposed workings proceeded simultaneously in more than two seams, and it has been demonstrated that simultaneous working of two seams is bad practice and liable to set up 'creep'. Under present practice, where more than one seam is worked, not more than one seam is worked at the same time over a given area.

The 'Main band' at Whitehaven ranges from 8 ft. to 12 ft. in thickness, all clean coal, and is worked entirely pillar-and-room.

The 'Bannock band', lying 120 feet above the Main band, is worked partly room-and-pillar and partly longwall. The seam is 7 feet thick, but where longwall is employed only 5 feet of the seam is taken out.

Earlier practice at Whitehaven was to drive wide rooms, and later to split up the pillars into four smaller ones, which were abandoned. Attempts have at various times been made to recover these smaller pillars, but with small success.

The Bannock band in the Wellington and William collieries at Whitehaven is worked over areas where, in the Main band below, the seam was formed into pillars and split into smaller pillars. The intervening strata is strong sandstone.

At the Harrington collieries, the inclination of the seams varies from 4 degrees to 11 degrees. The workings of the Harrington No. 10 colliery are $2\frac{3}{4}$ miles out from the shaft, or 9,000 feet directly seawards.

Several seams are worked under the sea, but no two seams are worked in the same area simultaneously.

Pumping is almost negligible in the submarine workings, which are, indeed, so dry as to require stone-dusting and removal of dust from the haulage-roads.

No shotfiring is done in the Main band workings. In the Bannock band, explosives are used only for ripping.

Ventilation.—The shafts at the Wellington colliery are very old, and are small. The coal-hoisting shaft, which is the downcast, is 9 ft. by 8 ft. diameter, of elliptical form. The upcast shaft is 12 ft. diameter. The main ventilator is a Walker fan, 26 ft. dia., passing 110,000 cu. ft. of air per minute at $6\frac{1}{2}$ inches water-gauge. Owing to excessive leakage, due to the long length of old workings traversed by the intake air, the water-gauge drops to one inch at a point $1\frac{1}{2}$ miles from the shaft. At this point a Sirocco fan, double-inlet and 50 inches diameter, is placed. This boosts the water-gauge to $2\frac{1}{2}$ inches, and passes 75,000 cu. ft. of air per minute.

The temperature of the most advanced faces is at the present time about 80 degrees F., the air being well saturated.

The difficulty of maintaining a tight airway through the 'broken' of a pillar-and-room area under the conditions of extreme length of air-travel necessitated by long-continued advance of the workings under the sea has been fully experienced at Whitehaven. A reorganization of the ventilation arrangements in the Wellington pit, by building sidewalls along the main air-road and installing a 'booster' fan inbye, had been almost carried to completion previously to the explosion and mine-fire of 1910. The necessity for an additional or alternative main-intake to serve remote submarine areas was raised by this disaster (1).

Transmission of power.—Electricity is used at Whitehaven for haulage, pumping, air-compression, ventilation, and junction-lighting underground. It is not used at the working faces, or at any point within 900 feet of the coal-face. compressed-air being here employed.

At the William pit, compressed-air is conveyed from the surface to a distance of 3½ miles 'inbye'. An electricallydriven auxiliary air-compressor is installed 21/2 miles inbye, of 600 ft, capacity, 80 lb. pressure, two-stage vertical type.

In the Wellington pit, inbye 2 miles, are two similar compressors of 500 ft. and 300 ft. capacity. Compressors, of similar type, are installed at distance of 1½ miles inbye in the Ladysmith pit, and at one-quarter mile in the Haig pit, of 2,000 ft. and 800 ft. capacity, respectively.

Electrically-operated auxiliary haulages are also used at distances exceeding 2 miles from the shafts (2).

The Royal Commission on Coal Supplies has estimated an extension of 12 miles seawards in the Cumberland field, the area lying between 5 miles and 12 miles from shore being classed as 'probable reserve'.

At Whitehaven the foreshore is owned by Lord Lonsdale, the Crown having sold it many years ago. Submarine mining is in this instance not subject to Crown-lease restrictions.

⁽¹⁾ Report on the Explosion and Underground Fire at the Wellington Pit, Whitehaven Colliery; British Home Office Blue Book, 1911 (Cd. 5524).
(2) Particulars of submarine operations, and the section of the workings at the Whitehaven collieries, have been kindly furnished by the General Manager, Mr. W. H. Johnson.

At the Harrington collieries, workings have extended for 2¾ miles from the shaft, 9,000 feet of this distance being under the sea. The maximum cover below the sea-floor is 750 feet, the lesser depth, compared with that of the Whitehaven workings, being due to greater upthrow of the seams by faults in the Harrington area. One seam 2 ft. 4 in. thick is worked by machine-cut longwall faces. The other seams are worked pillar-and-room. As at Whitehaven, no two seams are simultaneously worked over the same area.

The main haulages at both Whitehaven and Harrington are all endless ropes, running 3 miles per-hour when hauling coal and 5 m.p.h. when riding men. From $1\frac{1}{2}$ to $1\frac{3}{4}$ hours of the time spent by the men underground is taken up in getting to and returning from the working faces daily.

Northumberland and Durham Coast Fields:

As many of the early officials and workmen of the General Mining Association came to the Sydney collieries from the North Country coalfields of Great Britain, mining practice, and especially submarine mining-practice, in Nova Scotia has been much influenced by these early associations that are even today reflected in the local mining terms, and by such names in our Society's membership roll as Greenwell, Rutherford, Brown, etc.

The association of North Country place-names with undersea mining is noticeable in other instances, as the Newcastle seam under Nanaimo harbour, B.C., and the names of Hetton, Stockton, etc., given to collieries in the Newcastle area in New South Wales.

The coalfield of Durham and Northumberland is an irregular basin having a synclinal axis aligned north and south. The seams, in Northumberland, dip eastward under the sea, and, in Durham, are concealed along the coast by Permian rocks. Along the coast of north Durham and Northumberland, the dips going north change from a very slight eastward dip under the sea to an inclination of 1 in 10.3 (5½ degrees) and then to a rising inclination of 1 in 120.

In Durham, the general dip of the seams is seawards. The average inclination of seams followed under the sea is 1 in 16, gradually flattening out to about 1 in 85.

The field has a shore frontage of 46 miles, extending northwards from a point just north of Hartlepool.

An annual output of 4,100,000 tons is being produced from beneath the sea, approximately half of this quantity coming from north of the river Wear, and the other half from collieries lying between the rivers Wear and Tees.

The earliest undersea workings appear to have commenced at North Seaton colliery, in Northumberland, in 1872, and there appears to have been a general development of submarine mining about 1899 in south Durham.

Northumberland and Durham coast north of the Wear.—Coal has been mined in Northumberland for 4,500 feet, and in north Durham for 8,580 feet, seawards, with a maximum strata-cover below the sea-floor of 648 feet and 1,590 feet, respectively. The sea-bottom is very slightly inclined, falling 1 in 120 in Northumberland and 1 in 48 in North Durham.

The coal is mined under Crown leases, usually one seam at a time, and in some instances two seams.

In Northumberland, owing to diminishing cover seawards, the workable limit seawards under the lease terms has been reached.

In north Durham, men walk the whole distance from the shaft to the working faces, and this circumstance is acting as a present limit to extension of workings seawards.

Both longwall and pillar-and-stall methods are employed. South Durham Coast.—The collieries of Ryhope, Seaham, Dawdon, Easington, Horden, and Blackhall mine undersea coal between the Wear and the Tees (1).

The maximum seaward penetration in this district is 9,000 feet, and maximum undersea cover is 2,100 feet.

Both longwall and pillar-and-room methods are employed, with the last-named method predominating to date. All the coal is extracted in both methods. In nearly all cases three seams are being worked simultaneously under the sea.

⁽¹⁾ See article by J. L. Henrard in "Colliery Guardian" of 11th Feb., 1927, page 326, describing sinking in progress, using freezing method, at Seaham harbour. Two shafts are being sunk, passing through seven workable seams and winning the Busty seam at approximately 2,100 feet deep. The seams are overlain by the Magnesian Limestone and a running sand, both formations holding salt water which rises and falls with the tide and is in communication with the sea. The shafts are 250 yards from high-water mark, and will win undersea coal.

No difficulties have so far been encountered in extraction

of undersea coal in this field (1).

The estimate of the Royal Commission on Coal Supplies placed a three-mile limit on the undersea mining of coal along the Durham and Northumberland coast, but in the Durham area, at least, there is no reason to believe that three miles is the workable limit, as indications are that the coal-seams extend much further out to sea.

Full particulars of the undersea mining of coal in the Northumberland field will be found in a paper by Mr. T. E.

Forster, to which the reader is referred (2).

The following table of the composition of the strata overlying the Low Main seam at the North Seaton, Cambois, and Cowpen collieries is excerpted from Mr. Forster's paper for comparison with the table showing the composition of the strata over the Hub seam at Glace Bav.

	North Seaton Per cent	Cambois Per cent	Cowpen Per cent
Sandstones	42.17	56.64	45.34
Shales, and shales with thin sandstones		27.31	40.86
Coal-seams	4 50	4.25	3.73
Fireclays	5.08	11.80	10.07

Mr. Forster states that the sandstones pass water freely, as is the case in the Sydney field also (3).

Scotland

Firth of Forth:

The coalfield of the Lothians is connected with the Fifeshire coalfield under the firth of Forth. From the Fifeshire shore to that of Haddington and Edinburgh shires is nineteen miles. and of the continuity of the coal-seams across this distance there is no reasonable doubt (4). The whole series of coalbearing strata is present, and 130 square miles of coal-bearing measures are contained in a synclinal basin, having its axis north and south, with the seams outcropping under the firth on either side of this axis.

⁽¹⁾ For information regarding undersea mining of coal on the east and west coasts of England, the writer desires to thank Mr. T. Greenland Davies, H.M. Inspector of Mines for the Northern Division.

(2) "Undersea Coal of the Northumberland Coast", by T. E. Forster;

Trans. Inst. Min. Eng., 1903.

(3) R. H. Brown, Proc. Min. Soc. of N.S., 1904.

(4) British Royal Commission on Coal Supplies, 1905.

The axis of the basin swings towards the Fifeshire shore and there is a large portion of the basin where the lower seams are at a depth exceeding 4,000 feet.

The field is extensively worked from the Fifeshire shore. The Fife Coal Company's colliery at Leven, the Wellesley, Rosie and Michael collieries of the Wemyss Coal Company, and the Dysart colliery (Frances pit) of the Earl of Rosslyn's Collieries, Ltd., are all operating in undersea areas.

The Leven colliery works the 'Eight Foot seam' (5 feet thick), and the Chemiss seam (8 feet thick), entering the undersea area at 1,000 feet and 1,180 feet deep, respectively, the seams dipping seaward. Longwall method is used.

The collieries of the Weymss Coal Company work the same seams. The Wellesley colliery has workings extending 1½ miles from shore, which at this point are 1,800 feet deep. The Michael colliery, further east, is working under the sea at a depth of 1,800 feet at 4,000 feet from shore, the inclination of the seams increasing in a southwesterly direction from 1 in 5 at Wellesley colliery to 1 in 3 at Michael colliery (¹).

The Weymss Coal Company is engaged in developing an important new winning at the Michael colliery, East Weymss. A shaft 24 feet in diameter, sunk to 300 feet at the close of 1926, is intended to win the Dysart seam at a depth of 1,800 feet. The hoisting equipment is designed to lift 450 tons per hour, using cages holding twelve one-ton cars in two decks. The shaft will communicate at the level of the Dysart seam with workings in the Chemiss seam, and seams above, through a cross-measure drift, which, at the close of 1926, was being driven back from workings now at a depth of 1,800 feet and distant from shore 4,000 feet, previously referred to, the drift being driven simultaneously with the progress of the shaft sinking.

On the opposite shore, coal is being worked under the sea from Prestonlinks and Prestonpans coilieries, workings having proceeded beyond high-water mark for a distance in excess of 3,000 feet.

^{(1) &}quot;The Dysart, Wemyss & Leven Coalfield, Fifeshire", by Rich. Kirkby; Trans. I.M.E., 1902, vol. XXIII, page 291.

On this side, the coal-seams are cut by basaltic dykes which extend through the seams to the sea-floor, the coal being burnt on either side of the intrusions. In one instance, an inflow of salt water found its way into the workings apparently through crevices between the strata and the basalt dyke.

It is anticipated that at a future date the Fifeshire collieries will mine a distance of possibly eight miles from shore, and that the Lothian collieries will extend a similar distance. It is probable that mining operations will extend to the depth limit of mining along the slopes of the syncline, but will not be able to win the coal lying in the bottom of the basin (1).

The depth of the syncline is greater on the Fifeshire shore than on the Lothian shore, and while a portion of the upper series of seams only will be mineable from the Fifeshire side, it will be possible to mine the Upper, Middle and Limestone

seams from the opposite shore.

The depth of water under the Firth is mostly under 120 feet, and so far as known the sea-floor is covered with a thick deposit of stiff boulder-clay, a combination favourable to undersea mining. Two-thirds of the mineable coal under the firth will, it is believed, be mined from the Fifeshire shore, and one-third from the Lothians side (vide Report of J. S. Dixon, Royal Commission on Coal Supplies, 1905).

That undersea mining under the firth of Forth is as yet only in the initial stages, and that, as in Cape Breton, a second and more important stage is impending, is evident from the remarks of Mr. J. S. Dixon in his Report, dated 1904, to the

Royal Commission on Coal Supplies, who stated:

"The coal under the firth of Forth... above 4,000 ft. in depth is sufficient to maintain the whole present output of Scotland for upwards of sixty years. It will require to be worked by collieries on the shore, and also possibly, as has been proposed, on the island of Inchkeith. Many of the shafts will be sunk outside the Coal Measures, which will be tapped by long tunnels driven out to intercept the coal-seams at great depths. The workings will have to proceed from five to seven miles from the land to the limits of the coal seams. All this means large initial and working costs. The whole circumstances of this large submerged coalfield are such that it cannot produce an output commensurate with the large quantity of coal it contains, and its working on an extensive scale will only be begun when coal more easily accessible is getting scarce."

⁽¹⁾ Note: For particulars of mining under the firth of Forth, the writer is chiefly indebted to Mr. Richard Kirkby, of the Weymss Coal Company.

Borrowstounness or Bo'ness:

Upstream in the firth of Forth, twenty miles west of the eastern outcroppings of the Lothian-Fifeshire field, is the Bo'ness coalfield on the southern shore of the firth in Linlithgowshire.

Mr. Henry M. Cadell's paper in the Transactions of the Institution of Mining Engineers(1) gives a most interesting account of undersea coal-mining operations in this small field. which is one of the first, if not the first, contribution of this nature to mining literature.

At Bridgeness, the bed of the firth is covered first by "very hard, stiff, unstratified glacial 'till' or boulder clay, full of ice-worn stones and boulders", and above this is a bed of clay 30 to 40 feet thick, which is plastic and completely impervious to water.

After an unsuccessful attempt, in 1862, to sink a shaft at low-water mark by means of an embankment, and a further attempt in 1869, a cased-shaft, commenced in 1878 and completed in 1880, won coal at a depth of 340 feet in a site "surrounded by the sea" from which the winning of coal has successfully proceeded for many years. In one instance, coal was worked to its outcropping in the boulder-clay under the Firth, the clay being so hard that work was carried along for 150 feet with the clay as a roof "which was so hard and strong that its true character was not at first noticed" (2).

In this small field, coal appears to have been mined under tidal water with less cover than at any other point, because of the perfect nature of the clay mantle. It may be noted that Mr. Cadell was owner of the undersea coal by virtue of very ancient title and his mining plans were not subject to lease conditions. In three instances, seams have been worked to their outcrops against the boulder-clay at depths of from 137 feet to 400 feet below high-water mark.

^{(1) &}quot;Submarine Coal-Mining at Bridgeness, N.B.", by Henry M. Cadell; Trans. I. M. E., 1897-8, Vol. XIV, page 237.
(2) "At one place", Mr. Cadell stated, "an ice-worn boulder of hard brown sandstone about 3 feet in length, ellipsoidal in shape, splendidly streaked from end to end, rolled out of a hollow in the rock surface in which it had lain snugly buffed since the Glacial period, when the Forth valley was occupied by a huge glacier.'

The procedure of undersea mining under the firth of Forth is governed by covenants in the Crown leases, which are adapted to the special circumstances of each lease. Appendix 'C' is a tabular arrangement of regulations drawn to apply particularly to leases under the firth of Forth, condensed from clauses kindly supplied by Mr. Westgarth Forster Brown, the Commissioner of Woods and Forests.

The object of the exploring drift required in longwall workings is "to ascertain the position of any intersecting dykes, hitches, or troubles" and it is required that barriers of coal at least 30 feet in thickness "shall be left against all such dykes or troubles of which the throw or dislocation exceeds 30 feet or where the cheeks or sides are more than two feet apart."

Any issue of salt or brackish water must be immediately reported to the lessors.

Estuary of the Dee

An extension of the coalfield worked in Flintshire, north Wales, lies under the estuary of the Dee with boundaries not yet determined. Coal is being mined under the estuary at the Point of Ayr, Englefield, and Bettisfield collieries. Work at the Wirral colliery, on the north side of the estuary, has been abandoned during 1927.

Undeveloped Undersea Areas in Great Britain

The recently developed coalfield of Kent is believed to have a seaward extension under the straits of Dover along a shore frontage of some 20 miles, the probable workable limit of the extension being placed, for the estimating purposes of the Royal Commission, at four miles distant from and paralleling the shoreline.

The Kent coalfield has only just passed the pioneering stage of development, and the working of the undersea extension is a matter for the future. The coalfield is a concealed one, and the undersea extension is unique, inasmuch as the coal seams are at a considerable depth at the shore-line and entirely concealed by younger rocks. The existence of the Kent field,

which had been presumed by Godwin Austen as far back as 1855, was actually proved by a borehole at the foot of the Shakespeare cliff at Dover, where the Coal Measures were met at a depth of 1,157 feet from the surface.

Other probable undersea coal-areas are those under Swansea and Carmarthen bays, being extensions of the South Wales coalfield, estimated together at 20 square miles. No

mining has been attempted in either area.

AUSTRALIA

New South Wales:

The great coal-basin of New South Wales extends for 200 miles along the coastline of the South Pacific ocean. The deepest part of the basin is near Sydney, the measures rising to the north, south, and west boundaries of the field, and seawards. The extent of the submarine area is undetermined.

There are three main horizons of coal-bearing strata of

Permo-Carboniferous age, namely:

		Thickness
		in feet
1	. Upper, or Newcastle, series, containing 35 to 40 feet of	
_	workable coal	1,400 - 1,500
	Strata	
2	Middle, or East Maitland, series, with 18 feet of workable	
	coal	500 - 1,800
	Strata	
9	B. Lower, or Greta, series, with two thick seams, of varying	
	thickness, but averaging together 20 feet of workable coal	100 - 300
		. 4

The most important submarine development is connected with mining of the Borehole seam (8 ft. to 9 ft. thick), situated at the base of the Upper Measures in the Newcastle Delta area, in the neighbourhood of the estuary of the river Hunter and near the northern termination of the basin along the coast-line. This unusual instance of coal-mining under a combination of low-lying areas of sand and gravel, intersected by a tidal river, and under the sea, has been described by Mr. A. A. Atkinson in a classic paper, which, with the discussions that followed it, forms a chief part of the literature on submarine coal-mining (1).

^{(1) &}quot;Working coal under the River Hunter, the Pacific Ocean and its Tidal Waters, near Newcastle, N.S.W.", by A. A. Atkinson, Chief Inspector of Coal-Mines, New South Wales; Trans. Inst. Min. Eng., 1902.

The Newcastle Delta area has been worked over in very large measure to the cover limit of 120 feet.

The following acreages have been worked over, 38 per cent of the coal being recovered and 52 per cent left as pillars:

		area d over	Dista worki extend seawa	ngs ded
Hetton colliery	954 :	acres	7,590	feet
Stockton colliery		66	3,830	64
New Winning		66	1,120	66
Newcastle Coal Mining Co's "A" and "B" pits		44	1,850	66
Dudley colliery	174	64	4,290	66
	1,501 a	acres		

All the above-noted collieries are closed altogether or have exhausted their undersea territory.

Collieries now operating in this district are as follows:

	See area so far worked over	Distance workings extended seawards
Burwood colliery	. 131 acres	3,170 feet
Lambton	. 3 "	730 "

The greatest thickness of cover at the shore-line is only about 300 feet, and in several instances much less. The same worked — the Borehole seam — rises seawards, and the sea-floor falls, so that only a limited advance is possible before reaching the 120-feet cover-line.

The Report of the Department of Mines of New South Wales for 1916, stated in reference to the collieries named as having stopped undersea operations:

"In each case the extremity of the sea-workings is in good coal, frequently thicker and cleaner than it is nearer the land, and therefore it appears quite likely that in the future some enterprizing Company will come along and propose by efficient sand-packing to work more of this good coal — perhaps driving further seaward — but certainly recovering some of the pillar-coal left."

Mining of the Borehole seam near its outcropping under a cover of 65 feet, of which the upper 23 feet was sand and silt underlying a tidal creek, caused flooding of the Ferndale colliery in the Newcastle district in 1886, and loss of one life, an incident that led to the appointment of a Royal Commission and has had a great bearing on the subsequent practice of mining in New South Wales (1).

A special form of lease is used under the Mining Act for coal "under ocean, tidal waters, etc.", the principal provisions of which are excerpted in Appendix 'D'.

The special provisions of the New South Wales lease have quite evidently been suited to the unusual conditions they are intended to meet, as no subsequent inbreaks of water have occurred.

Coal has been mined under Sydney harbour for many years, but the operating companies have been hampered by the occurrence of igneous intrusions that were found to have destroyed the coal-seams, and by labour and financial troubles. The Balmain colliery is at this time mining coal under Sydney harbour at a depth of 2,886 feet below sea-level. The coal is completely extracted by longwall method. The workings have proceeded under water 1,980 feet from shore, and 55½ acres of undersea coal has been worked over(2).

Queensland

There is a development of coal-bearing rocks in Queensland in the vicinity of, and north of, Brisbane, and a possibility that at some future date coal-seams may be mined under the sea in the neighbourhood of Maryborough and Hervey bay, where the coal-bearing strata is of Cretaceous age.

Undersea mining of coal is, however, as yet a remote possibility in Queensland (3).

⁽¹⁾ Royal Commission on Collieries, Third Report on the condition of the collieries adjacent to Ferndale, Sydney; 1886, page 67. Mr. Atkinson's paper contains a summary of this Report.

(2) Information kindly furnished by the Under-Secretary of the Depart-

ment of Mines for New South Wales. (3) Information has been kindly furnished by the Department of Mines

at Brisbane, Queensland.

CHILE

Coal is being mined from submarine areas in Chile to the extent of 1,200,000 tons per year, one mine producing 350,000 tons, another 300,000 tons, and several between 200,000 and 300,000 tons annually.

The more important mines are situated near Coronel, and the main coalfield is under the water of Arauco bay in the direction of the island of Santa Maria. A marked feature of the field is the occurrence of numerous parallel faults, associated with mountain-building movements on the Pacific slope, which have generally speaking facilitated mining by bringing the coal-seams up to accessible depths. The seams occur with such relation to the coast-line that workings enter under the sea about at the shore-line, and in some instances access is obtained to upper seams outcropping under the sea by cross-measure tunnels as shown in Figure 17.

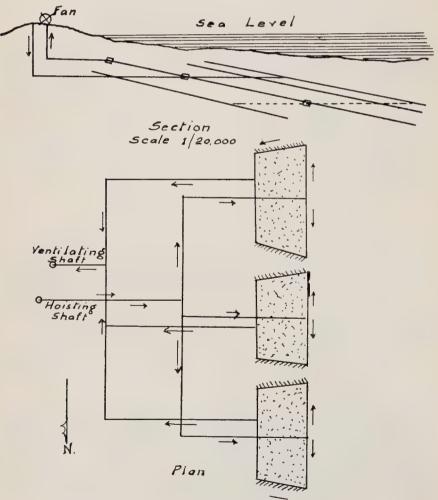


Figure 17.—Typical submarine winning in Chile, using shafts and cross-measure drifts. Separated longwall faces.

(From Report by Sr. Ed. Delcourt)

Workings have reached a distance of 8,000 feet in a straight line from shore. Some of the haulage and ventilation roads are 9,000 feet from shore, and are continually extending seawards.

The seams dip under the sea at an inclination of from 8 to 12 degrees, but the successive faults lessen the tendency of the seams to gain in depth of cover under the sea. The sea-floor has no appreciable inclination, being, at 12 miles seawards, from 90 to 150 feet below mean sea-level. An upper seam, which enters beneath the sea with 225 feet of cover at the shore-line, has attained a cover of 960 feet at a distance of 4,800 feet from shore.

The first mining, of date from 30 to 50 years ago, was by pillar-and-room extraction near the shore-line, with cover of only 120 to 250 feet between the upper seam worked and the sea-bottom.

A shale bed, 100 feet thick more or less, which acts as a seal, is present between the sandstone roof of the upper seam and the sea-floor.

The properties now worked by the Schwager Mining Company, the most important company engaged in coalmining in Chile, are at Puchoco, near Coronel. In 1881, the submarine workings of two colliery properties in this district were flooded by entrance of the sea. At this date the undersea coal was being won through shafts 600 feet deep sunk near the shore. It is believed that rise workings had been allowed to approach the sea-bottom too closely in the neighbourhood of a fault, which gave off a strong stream of water. No steps were taken to dam the flow, with the result that the gouge material along the hade of the fault was loosened and washed out, making free communication with the sea-bottom. flooding occurred on September 18th, a national holiday, and fortunately therefore resulted in no loss of life. The workings and shafts being connected, the whole operation, and that of an adjacent and communicating property, was abandoned. The coal-field was considered as irretrievably lost, until Mr. Graham, superintendent of a neighbouring colliery, suggested recovery of the seaward coal by means of tunnels to be driven in the sandstone measures overlying the coal seams. The late Gilbert Pitcairn Simpson was consulted on the proposed scheme, which was adopted, with some slight modifications in the plan suggested.

The tunnels were started above high-water mark and driven at a downward inclination of 1 in 4.5 for a distance of 2,400 feet, which brought them to a point above the line of the inundated workings. From this point the drifts were driven at an inclination of 1 in 3 until they touched the coal at a total distance of 3,800 feet from the starting point on shore.

The coalfield has been developed extensively from the intersection of the seam by the tunnels, a barrier of solid coal 300 feet in width being left as protection against the inflow of

water from the flooded workings (1).

The workings near the shore at the Playa Negra colliery near Coronel were flooded in 1876, and the coal to the dip had to be worked from a new opening.

Later mining of coal has been almost entirely by longwall

method.

A typical section of the coal-bearing measures in the Coronel district is as follows:

Strata	114 feet
Coal	3 ft. 9 in.
Strata	
Coal	3 ft. 11 in.
Strata	48 feet
	4 ft. 6 in.

The thickness of strata between the seams varies greatly within short distances, and there is evidence that the Coronel field, as well as others in Chile, were formed in separated basins of deposition (2).

From two to three seams of coal are worked under the sea simultaneously.

Very little water has been encountered in the workings in the more distant submarine areas now being worked, and in general such water as is met with is fresh water. 8,000 to 13,000 gallons per hour is made in typical submarine mines having a daily production of 1,000 tons. A maximum of 20,000 gallons per hour is made in mines having the largest inflow (3).

The submarine coal in Chile is the property of the State. and each submarine lease is the subject of a separate agreement with the persons seeking permission to mine coal, which must

Inst. Min. Eng., 1909.

(3) Particulars of the submarine conditions have been kindly furnished by the Director of the Bureau of Mines at Santiago.

⁽¹⁾ The writer is indebted to Mr. James Cumberford, of Amherst, for particulars of the flooding and recovery of the Puchoco area.

(2) "The Coalfields and Collieries of Chile", by Archibald Russell; Trans.

be sanctioned by special legislation. In 1889, a special law was passed entitling the foreshore owners to apply for 'pedimentos' or grants of the adjacent submarine coal. These grants being given are termed 'partenencias' or property rights. Since the date named, however, no further concessions for submarine coals have been granted.

There is no government regulation of mining operations. Mr. Rowland Gascoyne, writing in 1898, referred to longwall mining at the Schwager mines at Coronel, and noted this operation at that date as being the only instance of the use of this method under the sea in Chile, remarking that "had not this system been introduced there is no doubt that these mines would have been exhausted some time ago, owing to the pillars it would have been necessary to leave under the pillar-and-stall system" (1).

The successful continuation of longwall extraction at this time, thirty years later, and the greatly increased tonnage of output now being maintained is a significant circumstance.

With regard to the mining of superimposed seams, the writer is informed by Mr. James Cumberford that in the Schwager collieries two seams were worked with an intervening distance of 190 feet, and that difficulty was experienced in regard to maintenance, irrespective of which seam was first worked. With the upper seam worked in advance, the lower seam coming behind disturbed the roadways in the seam above, and the lower seam itself under these conditions was harder to mine. With the lower seam worked in advance, there was slight effect only upon the mining of the upper seam, but the roadways in the lower seam were disturbed.

Airways in the seams were found difficult to maintain, and it was found to be less expensive to cross-cut the measures and drive the main airways in sandstone rock. The heavy expense of these rock tunnels proved less than the cost of continual repairs on airways maintained in the goaf (2).

^{(1) &}quot;The Coalfields of Chile", by Rowland Gascoyne; Trans. Inst. Min. Eng., Vol. 15, 1898.

Eng., Vol. 15, 1898.

(²) Note: Ankylostomiasis, a disease caused by an intestinal worm (better known on this side of the Atlantic as hookworm disease) is prevalent among Chilean coal-miners. There has been no instance of infection of coal-mines in Canada. As the predisposing environment of combined heat and moisture is not present in Canadian coal-mines, development of the infection, if introduced, is improbable. If deep mining in submarine areas should be accompanied by high temperatures, the absence of moisture will prevent development of the environment necessary to propagation of the infection. See "Ankylostomiasis, 'Miner's Anæmia'", by F. W. Gray; Trans. Min. Soc. of N.S., 1906, vol. XI, pp. 75-97.

JAPAN

There are a number of occurrences of undersea coal in Japan, the more important being in northern Kyushu, the southermost island of Japan. The Chikuho field in this district is the most important coalfield in Japan, and has a submarine extension not yet worked.

Lying between Nagasaki and the naval base of Sasebo is the island of Matushima, where submarine mining is in progress, and north of Nagasaki is the Sakito field, where also there is undersea mining of coal. In both these instances, undersea operations are interfered with by inflow of water through cracks in the sandstone rocks, and through faults. The Matushima colliery produces 400,000 tons annually.

At the southern tip of the mainland of Japan, coal is being worked under the sea in the Onoda field, and future mining in the Ube section of this field will be entirely submarine.

The most important undersea mining operation in Japan is at Takashima. The coalfield here consists of a group of small hilly islands, built up of Tertiary shale and sandstone with coal. Most of the coal under the islands has been taken out, and mining is now entirely submarine (1). The whole areas is much faulted, and mining is difficult. The coal is of good quality.

The Takashima colliery of the Mitsubishi Company—annual output 300,000 tons—is a very interesting operation. It is situated on a small island of 120 acres in extent, which has been artificially built up for protection of the colliery works (see photograph, Figure 18).



Figure 18.—Hashima Pit, Takashima Colliery, Japan. Mitsubishi Mining Company.

⁽¹⁾ Kinosuki Inouye: Coal Resources of the World, page 343.

A section of the strata containing the coal seams which are being mined is as follows:

Upper 8-ft. seam
Strata
Goma seam 6 ft. 8 in.
Strata140 ft. 0 in.
Banto seam
Strata 20 ft. 0 in.
10-ft. seam
Strata120 ft. 0 in.
12-ft. seam

The strata consists of sandstones, shales and sandy shales. The seams are inclined at angles from 20 to 60 degrees. The sea-floor is flat, with an average depth of 100 feet.

The maximum depth of strata under which mining is taking place below the sea is 1,550 feet. The furthest distance of present workings from shore is 2,110 feet.

The seams are worked by advancing longwall-faces, the waste being fully packed with material obtained from rock-drifting and the roof. Over ninety per cent of the coal is being successfully won in undersea areas.

The minimum thickness of strata under which coal mining is permitted is 300 feet (1).

Apparently not more than one, or occasionally two, seams have been worked under the sea in the same area. A very large area of unproved submarine coal probably exists between the islands referred to, but its recovery will be attended with unusual difficulties.

At the northern point of the island of Formosa a coalfield exists with seams running parallel to the northwest coast of the Island and passing under the sea at the northern end. A small production is now being obtained from under the sea.

The comparatively small coal-resources of Japan, and the occurrence of so many of the coalfields near the coast, has made the undersea coal-mining a matter of unusual importance to Japanese mining engineers. The practice, while owing much to early advisers, chiefly British mining engineers, has developed an advanced technique suited to peculiar local conditions.

⁽¹⁾ These particulars are kindly furnished by the Mitsubishi Shoji Kaisha, Ltd., New York.

Not less than $1\frac{1}{2}$ million tons of coal is at this date being mined annually in Japan from undersea areas.

SPAIN

On the north coast of Spain, under the bay of Biscay, the Royal Asturian Mine Company mined coal in the Province of Asturias under the sea in a seam of varying thickness up to 39 feet, and having an inclination of from 17 to 25 degrees. The sea-floor is inclined at four degrees. At right angle to the coast, workings proceeded under the sea for a distance of 2,000 feet, and for 2,600 feet parallel with the shore-line. The strata-cover at the shore is 179 feet, and at 1,300 feet seawards the solid cover is 617 feet.

Mining has been practised here under the sea since 1833, and, until 1902, the whole of the seam was removed, and the waste packed. From 1902 to 1905 supporting pillars of coal were left. An inbreak of the sea occurred in 1905, since which

date mining has been confined to the land area (1).

This is the only instance of undersea mining of coal known to the writer on the mainland of Europe and Asia.

Acknowledgment is gratefully made of assistance received in the preparation of the foregoing paper from Mr. Walter Herd, Mining Engineer of the British Empire Steel Corporation, who read portions of the draft, and from his assistant, Mr. Alex. L. Hay. Also from Mr. Sidney C. Mifflen, who assisted with plans and data, and from Mr. J. J. McDougall.

Acknowledgment is made in the footnotes of the source

of other particulars recorded.

For courteous and general response to enquiries, the writer desires to express his thanks.

⁽¹⁾ The writer is indebted for these particulars to the Royal Asturian Mine Company per the British Consul at Madrid.

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APPENDIX A

COAL MINES REGULATION ACT OF NOVA SCOTIA

Clauses relating to Undersea Coal-mining

Section 34

As amended in 1924

Chapter 8. Section 34

Statutes of 1908

1. In the working of coal or stratified deposits in submarine areas, the following provisions shall apply:

- following provisions shall apply:

 (a) No submarine seam of coal or stratified deposits shall be wrought under a less cover than one hundred and eighty feet of solid measures; Provided that the owner or lessee of any such area may drive passage ways to win the mineral to be wrought under a less cover than one hundred and eighty feet, but not less than one hundred feet of solid measures;
- (b) A barrier of the mineral wrought of not less than fifty yards, twenty-five yards on both sides of the boundary lines of every lease, shall be left unwrought between the workings of every submarine seam;
- (c) Where there is less than 500 feet of solid measures overlying the seam or stratified deposit wrought, the workings of every such submarine area shall be laid off in districts of an area not greater than half of one square-mile, and the barrier enclosing each separate district shall not be less than thirty yards thick, and shall not be pierced by more than four passageways having a sectional area not greater than nine feet wide and six feet high; Provided that the Inspector may, if he deems it necessary, permit the said passageways to be driven with a cross section, not exceeding sixty square feet;

Unchanged

Unchanged

(c) Where coal is to be mined in any undersea or submarine area, the proposed system and plans of working shall before work is commenced be submitted in writing to, and be approved of in writing by, the Inspector; and no change shall be made in such approved system without the written sanction of the Inspector;

- (d) No district shall have its length when parallel to the general trend of the adjoining shore greater than one mile;
- (e) A proposed system of working the mineral in each submarine area shall before work is commenced be submitted to and approved of by the Inspector; and no change shall be made in such approved system without the written sanction of the Inspector;
- (f) The opening of a new lift or level in a mine already working in a submarine area shall be deemed the commencement of a new winning within the meaning of the section.
- 2. The owner, agent or manager of any mine to which this section applies, who contravenes or fails to comply with any provision of this section, shall each be liable to a penalty not exceeding one thousand dollars, and if offence complained of is continued or repeated after a written notice has been given by the Inspector to such owner, agent or manager, of any such offence having been committed, the Supreme Court, or a judge thereof, whether any other proceedings have or have not been taken, may, upon application the Attornevby General, prohibit by injunction the working of such mine; Provided that the Commissioner may waive or modify any of the provisions of the section when, on the report of the Inspector it appears to his satisfaction that valuable coal areas cannot be otherwise wrought or mined.

- (d) Where any workings are carried on upon the Longwall system, or where coal pillars are being extracted, or any total extraction of coal is being made in submarine workings, an exploring drift shall be driven at least one hundred and fifty feet in a seaward direction in advance of such workings;
- (e) In all undersea or submarine workings levellings, depth of cover and soundings shall be taken at least every three months and said levellings, depth of cover and soundings shall be marked on the plan of the workings, provided such undersea or submarine workings are under a less overhead cover than five hundred feet.

Unchanged.

Unchanged, with addition of words:

"or that the safety of the workmen will not be endangered by the mining of the areas in the manner proposed".

APPENDIX B

SUBMARINE COAL AREAS

Coal Mines Regulation Act of British Columbia

Mining Submarine Areas

No submarine seam of coal or stratified deposit thereof shall be wrought under a less cover than one hundred and eighty feet of solid measures: Provided that the owner, agent or lessee of any submarine coal area may drive passage-ways to win the coal to be wrought under a less cover than one hundred and eighty feet, but not less than one hundred feet of solid measures unless the condition of strata overlying such proposed passage-ways warrants the Minister, on the written report of the Chief Inspector, permitting a lesser cover than one hundred feet of solid measures; and provided further that nothing herein contained shall prevent any owner, agent, or lessee from winning water-covered coal areas, where other means of access thereto are not available, by caisson, shaft, or concrete revetment, or by any safe method whereby any shaft or opening may be safely and securely sunk or driven and maintained through such water-covered areas; but any coal or stratified deposit so won shall be mined and operated subject to the provisions of this section; and provided further that the Minister may grant such exemption from the provisions of this section to mines already opened, under such conditions as he may deem safe. R.S. 1911, c. 160, s. 27.

28. Before commencing to mine any coal or stratified deposit thereof in a submarine coal area, the owner, agent, or lessee shall submit to the Chief Inspector a plan of the system whereby the submarine coal area is proposed to be worked, and the system must receive the written approval of the Chief Inspector before mining operations are commenced, and no change shall be made in such approved system without the written consent of the Chief Inspector. R.S. 1911, c. 160, s. 28.

The mine-plan of all submarine coal areas shall show the depth of solid cover at specified distances along the lines of all main roads and around the working-faces, and soundings shall also be taken at reasonable distances and recorded on such plan; and it shall be incumbent upon the owner, agent, or manager to furnish to the Chief Inspector or to the Inspector for the district, when required and if reasonably practicable, the depth of any marine or alluvial deposit of sand, mud, silt, gravel, or drift which may overlay any submaring coal area in which mining operations are carried on a rare intended. submarine coal area in which mining operations are carried on, or are intended to be carried on, by the owner, agent, lessee, or manager. R.S. 1911, c. 160, s. 29.

APPENDIX C

TABULAR ARRANGEMENT OF CONDITIONS TO BE OBSERVED UNDER CROWN LEASES AT THE FIRTH OF FORTH

Specification of methods at Working permitted at stated depths of Under-sea Strata Cover.

COMMENT AGENCY REPORT		DEPTHS OF COVER	3 COVER	
OF EXTRACTION	Seams four feet thick and under	Seams 4 ft. to 4 ft. 6 in.	Seams 4 ft. 6 in. to 7 ft. 6 in.	Seams 7 ft. 6 in. and over
Pillar and Stall', without power to remove pillars and agreement upon size of pillars to be left	270 ft. to 360 ft.	270 ft. to 480 ft.	270 ft. to 810 ft. 1,080 ft. and over	1,080 ft. and over
'Pillar and Stall', with power to remove pillars, but with packing of the waste	360 ft. to 810 ft.	480 ft. to 810 ft.	810 ft. to 1,080 ft.	:
Longwall, with packed waste (1)	360 ft. to 810 ft.	480 ft. to 810 ft.	810 ft. to 1,080 ft.	•
'Pillar and Stall', without restriction	810 ft. and over	810 ft. and over	1,080 ft. and over $1,080$ ft. and over	1,080 ft. and over
Longwall, without restriction	810 ft. and over	810 ft. and over	1,080 ft. and over 1,080 ft. and over	(2) 1,080 ft. and over

Notes: (1) In Longwall workings where the ground has not already been proved by workings in another seam, an exploring drift must be driven 150 ft. in advance of main workings.

(2) Lessors may object if in their opinion danger exists.

2 E

Only with express permission, and fulfillment of any special conditions that Lessors may specify.

APPENDIX D

Excerpt from Coal Mining Lease of the State of New South Wales, Australia, "Under Ocean, Tidal Waters, etc." under the Mining Act, 1906.

- (a) The minimum width of pillars to be left shall be eight (8) yards, and the maximum width of bords or other excavations shall be six (6) yards but the lessee may at his discretion increase the minimum width of the pillars or decrease the maximum width of the bords or other excavations so as to increase the general safety.
- (b) The eight (8) yard pillars shall be left unwrought.
- (c) All headings and bords shall be driven by plumb lines.
- (d) All coal workings shall be accurately surveyed every three months, the plan thereof shall show the area worked out during the previous three months, and every year's workings shall be indicated thereon by some distinctive colour. The dates of such survey shall be noted on the plan.
- (e) The colliery plans shall contain a faithful and accurate record of all dykes, faults, fissures and occurences that are met with in the mine, and the workings shall be delineated thereon as they are and not as they are intended to be.
- (f) In one road of every pair of winning off or leading headings a bore shall be kept going ten (10) feet in advance for the purpose of foretelling the presence of any fissure, wash-out, open-joint, fault, dyke, or otherwise and all winning-off headings shall be driven at least one hundred (100) yards in advance of the working bords.
- (g) The lessee shall on discovering a fault, wash-out, dyke or fissure in a bore or otherwise at the face or side of the leading headings or levels take all necessary precautions against possible danger before opening up such heading or level by the drive. The Chief Inspector of Coal Mines may require the lessee to leave coal unworked next to faults, dykes, etc. wherever he may consider it necessary.
- (h) Before commencing to work the coal under the ocean or under any river, lake, or estuary, the lessee shall notify the Secretary for Mines of his intention so to do. In addition to the special facilities provided for the escape of the men by the shafts the coal under the ocean, river, lake, or estuary, should not be attacked until after a large goaf has if possible been made by extensive coal workings under the land.
- (i) The most accurate and trustworthy information shall be obtained by the lessee not only of the depth and character of the ocean bed, river, lake, or estuary deposit but also of the strata overlying the coal seam. When working under the ocean the strata shall be bored through and proved a minimum thickness of thirty (30) feet at the face of the leading headings or levels immediately such headings or levels are under the ocean and after the first borehole has been completed other boreholes shall be so put up in advance of it at the face of the headings or levels at distances not exceeding twenty (20) yards.
- (j) In all workings under the ocean or under any river, lake, estuary or tidal waters, no coal shall be worked with less than one hundred and twenty (120) feet of good sound strata between the bed of the ocean, river, lake, estuary or tidal waters and the coal seam and it shall be the duty of the lessee to prove that the said thickness of cover exsits and to ensure that it is not diminished.

APPENDIX E

Notes on Ventilation of Coal-Mines Working Undersea Areas in the Sydney Coalfield, Cape Breton Island

	COLLIERIES						
	Dominion Coal Company, Limited Nova Scotia Steel & Coal Co., Limited					& Coal	
	1B	2	4	12	14	Prin- cess	Flor- ence
Maximum distance of workings from shaft in feet Maximum dis-		12,300	15,000	7,400	7,800	13,000	12,200
tance of workings from shore, in feet	12,600	8,100	9,600	4,500	6,400	12,400	8,600
Number of Main Intakes	Two	Three	One	Two	Two	Two	Two
Dimensions, in feet	18 x 7 9 x 6½	10 x 6½	11 x 6½	11 x 5½	11 x 5½	12 x 6	12 x 6
Type of fan	Sirocco	Walker	Sirocco	Walker	Walker	Capell	Capell
Size of fan, feet.	5 x 10 Forcing	20 x 7½ Forcing	7 x 3½ Forcing	11 x 4½ Forcing	11 x 4½ Forcing	20 x 5 Exhaust	15 x 7 Exhaust
Quantity of air circulated, in cubic feet per mi-							
nute	105,000	167,000	58,000	92,000	81,000	51,000	65,000
inches	3.55	4.5	3.0	2.0	2.85	4.5	2.6
flammable gas in main returns Quantity of air circulated, in cu. ft. per minute per	0.60%	0.40%	0.60%	0.40%	0.45%	0.50%	0.40%
worker under- ground Face temperature, degrees Fahren-	200	257	210	245	284	127	162
heit	51	65	57	63	61	57	57
working faces Average daily	94%	89%	90%			85%	90%
output, in tons Thickness of seam	2,745 7 ft.	3,500 7 ft.2 in.	2,000 6 ft.7 in.	1,100 6½ ft.	1,150 6½ ft.	1,350 5½ ft.	1,800 5 ft.

Notes: Electrical drives are used on all fans, except at collieries Nos. 2 and 4, where steam drive is employed.

A steam-driven standby fan is provided at all collieries except No. 14. At the Princess colliery, an auxiliary 'booster' fan is installed on the main intake, 8,800 feet from the intake shaft. This fan circulates 55,000 cu. ft. of air per minute against a water gauge of 1.9 inches, making the total resistance for the mine equal to 6.4 inches w.g.

APPENDIX F
Dates of Entrance of Colliery Workings into the
Undersea Extension of the Sydney Coalfield

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INTRODUCTORY REMARKS AND DISCUSSION*

MR. F. W. GRAY: It is not my intention to read my paper as it is too long, so I will touch only on the high lights. The Sydney field is the most favourable example of undersea coal mining that exists, and I have not been able to find anything like it elsewhere. The nearest approach to it is a field in England where the Whitehaven collieries are operating, but it has a lot of faults. The net result of these faults so far met with in the seaward advance of the workings has been to increase, by something over 100 feet, the gain in depth of cover, in a distance of approximately three and three-quarter miles.

In the Sydney field there are no faults of any consequence. On the east coast of England the submarine coal-seams rise going seawards, and mining is thereby limited. In Australia the sea bottom falls and the cover becomes too light. On Vancouver Island the coal ran out, and in Japan the seams are steep and contorted. The Chilian fields resemble our own to some

^{*}The discussion, pp. 121-182, took place when Mr. Gray's paper was presented at the Annual Meeting of the Mining Society of Nova Scotia, at Baddeck, C.B., June 21st-22nd, 1927.

extent. An inundation of two collieries took place there some time ago, due, it is said, to the rise workings being allowed to approach the sea bottom too closely in the vicinity of a fault. The Cape Breton undersea field has been proved over a frontage of about thirty miles and has been attacked and penetrated for a considerable distance along that front.

We have passed the prospecting stage and now generally use developed land areas as points of attack to win the undersea coal. Of our mining under the sea, it can be said that, with the exception of No. 1-B and Princess colliery, Sydney Mines, no real attempt has been made to win the undersea coal.

The mining practices of the future must be an effort to minimize distance, and while there are many factors to be considered, these are all subordinate to the problem of ventilation. There must be more ample openings than we have been accustomed to at the shore line, and the main airways underground must also be adequate. Horizontal or cross-measure drifts will have to be used in an attempt to minimize distance.

Another question is the transmission of power, and, not so much the kind of power as how it can be transmitted, is the important item of submarine mining. To transport power over long distances without loss of efficiency, it seems that electricity is the only thing available, but it is not advisable to use electricity at the coal face. However, the trend of years to come is towards an increased use of electricity underground and the development of flame-proof motors. Consideration will also be given to the generation of motive power 'in-bye' in order to avoid long pipe lines, etc. There is also the question of haulages of the required swiftness and of large capacity, but these, as previously stated, are subordinate to ventilation.

The problem of a more complete extraction of the coal not only refers to the importance of holding up the roof and securing complete subsidence, but also comprises the factor of remote distance. How long the cost will permit continuing to go to sea, leaving fifty-five per cent of the coal in the mine, is difficult to tell.

There is also the question of a better understanding of the geological conditions of the field, and improved geological maps. I submitted my paper to Dr. Camsell, and the presence of Dr. Bell and Mr. Norman is an indication of the interest the Geological Survey is taking in the problem. In 1873, Mr. Robb studied in a preliminary manner the question of folding or faulting under Sydney harbour, promising further study, but no further work has been done in this special locality. This is not as it should be, as the Sydney area is, and always has been, the biggest producing field in the Dominion.

I have attempted to indicate on the diagrammatic plan, included in my paper, the geological structure of the Sydney coalfield with the hope that, if it was not exactly right, the Geological Survey would take cognizance of it and make any necessary corrections. It has been mentioned by Dr. Bell and Dr. Hayes that seams found in the southern part of the Sydney coalfield are not continuous over the whole area, and while I have tried to deal with this phase to some extent, it is a special study in itself. Twenty-five per cent of the entire coal output in all Canada comes from under the sea; fifty-five per cent of the total output of Nova Scotia, and one hundred per cent of the output from the Sydney Mines district. Consequently, it is time to collect data and develop specialists in undersea mining.

To understand the problem of the generation, and of transmission, of power for submarine workings, it will be necessary to direct our thought along the lines of our own especial requirements. The conditions of submarine mining are unusual and we have, in the Sydney field, possibly a classic example of undersea mining for the whole world. In my paper I have not attempted to draw general conclusions, as these will have to be built up and tried out as we go along. I have been more of an observer than a participant in coal-mining operations, but hope that my paper will prove interesting and that from it something may crystallize that will help to solve our My paper will be up for discussion before the Mining and Metallurgical Congress to be held this fall, and it may have to be revised, and I hope that something may be condensed from this meeting to assist in the revision; that is why it will be discussed here today.

MR. W. HERD: I have listened with interest to Mr. Gray's introductory remarks. His paper, to my mind, is one of the most valuable ever presented to the Mining Society of Nova Scotia, and is particularly valuable to those living in Cape Breton, as our bread and butter principally depends on the success of submarine mining. The future of the Dominion Coal Company and the Nova Scotia Steel and Coal Company is bound up with the problem of extracting coal from under the sea at a cost that will permit a reasonable return on investment, and, whatever methods are adopted, the question of the cost of mining must enter into any scheme that may be devised. This important item enters not only into the present operations, but the winning of super-imposed seams as well.

It will now be in order to discuss Mr. Gray's paper, and those who had an opportunity to study it and prepare written discussion can first present their views.

MR. J. C. NICHOLSON: All who have read Mr. Grav's paper will agree that it is a most valuable contribution to the Mining Society of Nova Scotia, and will realize that only a man of Mr. Gray's calibre could prepare such a paper. The copy of it which I received was a little late coming, so I did not have time to study it thoroughly. There is, however, one phase of the difficulties encountered in submarine mining on which I can make a few remarks, and that is in connection with the transportation of men to the working faces. At the present time in our deeper mines the time spent travelling to the faces is from three-quarters of an hour to one and one-half hours, which means that the time at the working face is reduced to the extent mentioned. Representations have been made to the Government for an eight-hour day from bank to bank, but fortunately those in charge could see that it would be a dangerous proposition and would make it so difficult for the Companies, that, in all probability, they would have to discontinue operations.

What the situation will be in twenty years' time gives one food for thought. Mr. Gray spoke of No. 1-B being in operation two hundred years hence, but none of us will likely be around at that time. The Phalen seam in the Glace Bay district and the Sydney Mines collieries are going rapidly seaward, and in five or six years' time will be so far away that

better facilities will have to be provided for transportation. We have rakes at the present time, but their speed is comparatively slow—500 or 600 feet a minute—so that too much time will be taken up in travelling three or four miles at that rate of speed. Appliances for speeding-up transportation to a rate something like that of an express train will have to be looked into. This will mean improving tracks, rope haulage, and, if motors are to be used, consideration will have to be given to the grades. This is one of the problems that will have to be seriously looked into, and not only transportation of men, but the transportation of coal, will bear scrutiny.

The fitting up of a mine like No. 1-B means money, and the problem is to find people who will put up the money. Coal mining, at its best, is a gamble, but fortunately some people can be found to take a chance. Someone has said that, in the future, mine workers will live underground, but I think that is remote. Building an island in the sea has been proposed as a means of facilitating transportation in submarine mining, but I don't know if such a thing would be feasible off the Atlantic coast, considering the prevalence of storms and the great quantities of ice that block our shores in winter. We have already Flint island, off No. 6, which perhaps, some say, will be utilized, and perhaps an island can be built off Northern head, Lingan.

MR. J. R. DINN: Mr. Gray outlined the early history of mining in Nova Scotia, about one hundred years ago. We were not thought of then and will be forgotten in two hundred years when No. 1-B is finished. He spoke of the wasteful methods of mining in the old days, for which some of us suffer now. Then, in some cases, all the pillars were taken out, and it devolved on us to build-in new ones in order to maintain the collieries for present production.

This morning in his Presidential address, Mr. Gray referred to the reduction of productive men at the present time compared to a few years ago. I would like to point out to Mr. Gray something that may have escaped his attention, although I must say that few things do, and that is, the methods of the productive men have changed. He did not mention the tons per man from producers, which have increased through

the change in system. The practice of cutting, shooting, and loading, which largely replaces the old method of undercutting with shooters and loaders, has decreased the number of productive men, but under the new arrangement the same output is obtained with 20 or 25 less men than could be produced under the old way.

Mr. Nicholson made a good point when he referred to the problem of transporting men to the face. Those who will have charge of the mines during the next thirty years will be faced with the problem of getting men to their place of work and also of maintaining the average number of hours at the face. The men want to get to work earlier and home earlier, but the difficulty is to get them there. This will be one of the biggest problems to solve, as there will be a limit to the speed going and coming. This factor, as everybody knows, will tend to lessen production.

Mr. Gray referred to the question of power for production, and in this connection it might be mentioned that a great effort has been made during the last few years to extend large pipe-lines. It is hard to tell where all this will end. If a colliery is continued in operation for two hundred years and the mine is not electrically equipped, the power question will be a very difficult one indeed. In fact, the only way I see that power can be provided is by electricity. The matter of pumping air for the economical mining of coal is a subject that will require deep thought and very much consideration.

MR. A. McEachern: Having read Mr. Gray's most valuable contribution on undersea coal mining in Nova Scotia, I have no hesitation in stating that the paper is more comprehensive and far-reaching than anything I know of. In his paper, however, he has offered unfavourable criticism of the findings of the Commission on the flooding of the Port Hood mine. I was a member of that Commission and associated with me were Messrs. Hiram Donkin, one of the best engineers in Canada, who for some years was General Manager of the Dominion Coal Company, and afterwards Deputy Minister of Mines for the Province; also Norman McKenzie, the present Deputy Minister of Mines, one of our best mining men, who has a wide experience of mining.

Mr. Gray also refers to the flooding of a mine at Mabou. This mine entered the undersea area at a cover of 110 feet A level was driven at right angles to the slope by which the water found its way into the mine. About 90 gallons per minute came in. Some time after this the Port Hood mine was inundated, which brought about the appointment of the Commission referred to.

MR. W. HERD: Will Mr. McEachern describe the conditions at the Port Hood mine for us, so that all can understand the situation there.

MR. ALEX. McEachern: The Port Hood slope dips at an angle of about 20 degrees, getting easier at the lower end The slope is a little over 3,000 feet long. The outcrop is about 2,000 feet inland from tide water. The cover at the point where the slope enters under the sea is about 50 feet. The strata are faulted on the south and also on the west, and are thrown up at a high angle. There is 942 feet of solid measures, composed of 874 feet of sandstone, 18 feet sand, and 40 feet of shale, overlying the coal. The sandstone can be traced along the shore, and what we saw of it was badly fractured. There is a pond some little distance from the shore fed by a fresh-water brook. In very dry weather the brook has been known to dry up. The men of Port Hood claimed that the flooding of the mine was due to a crush on No. 2, which caused the water of the pond to break through and run down along the line of cleavage to the place where the pillars were extracted on No. 5 landing. We could find no evidence to substantiate the alleged statement of a crush. All we could find was that the said crushed district took only sixteen shifts to repair; neither was there any evidence to prove that the pillars were robbed, extracted, or even disturbed, where the crush was said to have taken place.

The reference in Mr. Gray's paper to the work of the Port Hood Commission, while not intended to be unfair, does them an injustice. Knowing Mr. Gray as we do, we cannot believe that he would write anything but what he considers to be reliable, and based on good judgment. In this case, however, in dealing with the Report of the Commission, he has read into it things never said, or intended to be said, by them. He also seems to have formed his own conclusions on

false premises.

Mr. Gray states: "The writer is unable to agree with the conclusion of the report regarding a vertical break in the strata, because this involves admission that extraction of a single pillar in a seam under seven feet thick could, without great lapse of time, cause a vertical crack of the strata extending 942 feet to the sea bottom, whereas much more extended pillar drawing of much earlier date, at a point nearer the sea bottom, was not followed by any inflow of water into the coal seam itself."

The Commission never said there was a break caused by drawing a pillar. What we did contend was that sea-water descended more or less vertically through some of the many fractures in the strata, and when six pillars instead of one were drawn for a distance upward of 375 and 175 along a barrier pillar left for support, the water found its way into the mine.

Even after making a thorough investigation into a creep or crush, which the workmen allege took place at No. 2, letting the pond water into the mine, and where, the paper states, "there was more extended pillar drawing of much earlier date at a point nearer the sea bottom", we failed to find any evidence to convince us that there was any movement contributing to the flooding of the mine. This is the point at which Mr. Gray states that the draw of the roof so opened up the overlying measures as to cause an accumulation of sea water in the strata, fed by gradual percolation of sea water down the jointing of the metals and gaining original admission where the strata planes crop out at the sea bottom. No pillars were drawn here or at any other place in the mine, except at No. 5, where the inflow of water occurred.

It was this idea we had in mind when we stated that "the mine is flooded with the water of the ocean, and we are more inclined to believe that it was through a fissure in the rock more nearly vertical in direction, than that the water followed the strata a long distance, finally making its appearance in the mine at the point above described".

After making observations of the sandstone strata on the shore, which seems to be badly fractured, and informing ourselves of the geological features of the strata adjacent to and overlying the Port Hood coal seam, including the faulting, the thickness and nature of the sandstone and shale, also the

depth of sand on sea bottom, and after reviewing the evidence of all who best knew the mine workings of the past, up until the mine flooded, we formed the conclusions of our report.

Mr. Gray looks on these conclusions as negative, but to us they have a very positive side. We both agree that the sea water got into the mine and filled it. How it got in, is the point of difference. Seeing that no pillars were drawn above No. 5, as already stated; that a sufficiently strong barrier pillar had been left in; that there was no evidence to substantiate any alleged creep; and that water only broke in after pillar-drawing began; we could arrive at no other conclusion than that the forty feet of falling shale below the fractured sandstone loosened and let water in. These are surely consistent reasons and are based on the knowledge we gathered at that time. We have had no occasion since then to change our theory, for that it only is, and must be, until proved otherwise.

We thought it advisable to make this statement which we know will add to Mr. Gray's information and place the matter in a different light than he saw it. There was much evidence taken, but unfortunately it was not given to the public.

MR. HERD: I understand, Mr. McEachern, that the water in the Port Hood mine rose and fell with the tide.

MR. McEachern: We found that the water in the mine rose and fell in unison with the tide, but not to the same degree. In the mine we found a difference between high and low water mark of 0.82 feet, and at the shore of 3.14 feet. As Mr. Gray says in his paper, the water in the mine is not falling or raising now. The break is 1,100 feet outside of tide water and we thought it more or less vertical on account of the appearance of the sandstone on the sea shore.

MR. GRAY: I do not want anything to get by that is not correct. If anyone can throw any light on this mystery, now is the time to do so.

MR. HERD: As Mr. Gray says, there are some very disquieting features about the Port Hood incident. He has given an instance where inundation occurred with about 125 feet of cover, but there are no records of this taking place where the cover was anything approaching 900 feet.

MR. RORISON: I talked with miners who worked there and they said that pillars were robbed.

- MR. H. A. McLeod: I was at Port Hood for about two years and during the time I was there no pillars were robbed or drawn. I talked with the underground manager who was then there, and he informed me that no pillars had been drawn.
- MR. J. J. McDougall: I know nothing of Port Hood mine previous to its being flooded, but I made a report on the property in 1921 and 1922. In the course of my investigation I ascertained elevations of water underground by barometric readings, and I satisfied myself that the water not only stood at sea level but rose and fell in unison with the tide.
- MR. H. A. McLeod: The information one could get from the people in the vicinity was conflicting, but I was told that there was an old pit called the 'American mine' from which the coal was drawn in a haphazard manner, and these old workings got connected up with the Port Hood pit that was flooded. The pillars in the American mine were drawn, and this is why the information was conflicting. There was no connection underground between the pits.

MR. NORMAN McKenzie: I do not think that very much information can be obtained by discussing the flooding of Port Hood mine at this date, as this mine was flooded in June, 1911, and no doubt any of the members present who knew anything about the accident at that time have forgotten the details during the intervening 16 years. The members of the Port Hood Commission present have forgotten very much of the details.

If you turn to page 50 and the fourth paragraph of Mr. Gray's paper you will find he states: "With regard to the flooding of the Port Hood mine, the writer believes that the whole story has not transpired". This may mean that it is not the whole story of the flooding of the mine, which may be very true or it may mean that the story told by the Commission is not a true one. The Commission decided the course of the water entering the mine as coming from the ocean, and, from the evidence, that it came from the roof after a fall had

taken place in the pillar section. The Commission reports: "Our conclusion is that the mine is flooded by the water of the ocean, and we are more inclined to believe that it was through a fissure in the rock more vertical in direction than that the water followed the strata (along the bedding) a long distance finally making its appearance in the mine at the point above described. We have given the source of the water, the course it followed between these two points. From the evidence and a study of conditions this is the most reasonable conclusion we could arrive at". I have no desire to deprive the writer of Undersea Coal Mining of his opinion in this matter as to how the mine was flooded. He has as much right to have an opinion in the matter as any member of the Commission, but I do object to his misquoting the Commission's report. On page 50, paragraph 5, Mr. Gray states: "If the Port Hood mine was flooded through a vertical break it is a unique instance. It is the writer's belief that it was not so flooded". The Commission's report does not say that the water came through a vertical break, but that it came through a fissure in the rock more nearly vertical in direction than if it followed the lines of bedding for a distance of over 2,000 feet from a point near the sea-shore.

Mr. Gray's opinion is on page 75, paragraph 4, which is as follows: "The suggestion of the writer, made at the time of the investigation, is that the draw of the roof above the extracted pillar-area in the third level so opened up the overlying measures as to cause an accumulation of water in the strata, fed by gradual percolation of sea-water down the jointing of the metals, and gaining original admission where the strata planes crop out in the sea-bottom. The drawing of a pillar at what was virtually the deepest place in the mine, and therefore exposed to the greatest head of pressure from any water contained in the strata, occasioned, it is surmised, a break-through of a large volume of salt-water indirectly connected with the sea—probably between high and low-water marks—and presumably controllable with adequate pumps".

Now suppose that I agree with the writer of Undersea Coal Mining and admit that pillars were extracted on No. 3 lift and the roof came down causing this draw, would it not be reasonable to expect that the water would drain in the

mine at No. 3 instead of following the bedding down to No. 5 over a solid portion of the roof which had not fallen? I have no knowledge of the pillars being extracted on No. 3 lift. According to the evidence no pillars had been extracted in this mine except the place where the water came through and flooded the mine.

My answer to the 3rd paragraph on page 75, is that the leak in the roof occurred on Thursday, June 22nd, 1911, and the Commission started their work April 18th, 1912, ten months after the accident. The mine was filled with water as it is today, consequently, we had no opportunity of visiting the mine to get first-hand information; but we have the sworn testimony of about 38 men who knew the condition, and most of them worked in the mine in some capacity, from the men who were in charge to the men who worked in the pillars and first noticed the water coming from the roof.

We had various theories presented to us as to the source of water, and examined each suggestion. One suggestion was that the water came from the pond on the beach near the shore line. This pond stood higher than high-water. Another was that it came from the little river. We examined this and found it contained little or no salt compared with the water in the mine. This place was also above high-water.

We then examined as much of the structure of overburden as we could see, also the direction of any faults, fractures. and disturbances of any kind, comparing the same with any indicated on the mine plan. We have no evidence to show any large fault in the pillars where the water came through. The men working the pillars said that they had lypes in the coal but could not notice very much displacement in the roof or pavement. One of the largest displacements on the plan. if projected, would intersect at a point between Nos. 2 and 3 rooms where the pillars were extracted. This is the most reasonable suggestion that I can see the Commission could give that the dome of a fall made by the extraction of six or seven pillars with the width of that many bords would reach a fissure making a connection to bottom of the sea. This is quite obvious with indications of a displacement of that nature being in the vicinity of the extracted pillars.

If we accept Mr. Gray's suggestion there is grave danger of extracting pillars under the sea or along the shore if there is a connection along the line of bedding from the bottom of the sea to the mine. This would make mining undersea and along the sea-shore more hazardous than most of us think it is.

I would like to quote briefly the statements made by the writer of Undersea Coal Mining in connection with the flooding of Port Hood mine, which are as follows: "The whole story has not transpired". "If the Port Hood mine was flooded through a vertical break it is a unique instance. It is the writer's belief that it was not so flooded." He then refers his readers to page 75 where he makes further statements on the subject: "Rather extensive pillar-drawing had taken place in a higher level in the mine at a date much earlier than the inbreak". "With regard to the actual cause of the flooding, the conclusions of the Report are negative". "The writer is unable to agree with the conclusions of the Report regarding a vertical break of the strata, because this involves admission that extraction of a single pillar in a seam under seven feet thick, could, without great lapse of time, cause a vertical crack of the strata extending 942 feet to the sea bottom, whereas much more extended pillar drawing, of much earlier date, at a point nearer the sea-bottom, was not followed by any inflow of water into the coal-seam itself". I feel sure that if Mr. Gray had been in possession of all the facts and knowledge the Commission secured at the investigation he would have taken a different view and have written otherwise.

I think if Mr. Gray takes the time to study the structure, plans, profile, and evidence, he will find that the story has been fairly well told by the Commission.

MR. J. J. McDougall: I seek this opportunity to congratulate the Society on being presented with such an excellent paper from Mr. Gray. It is a combination of history, facts, and original ideas. Ideas, I may say, on which we may build, and hope to take advantage of, as in a short time we shall have to go into submarine mining extensively. The paper is not a finished article but points out several things that will have to be considered carefully.

Mr. Gray is rather hard on the engineers of the past. His criticism, I think, is undeserved, as they were embarrassed as much as we are, and had just as hard a time to keep above water; so they evidently decided to let the future look after itself.

Princess mine was opened about fifty years ago, and if an expenditure had been made then on a par with that at 1-B today, our difficulties would be lightened. In the old days there was no compressed air and no electric power, and they lacked many other advantages that we enjoy. The distance proposed seaward was one mile, which was a big undertaking, as one mile then was the equivalent of four miles at the present time.

MR. JOHN MOFFATT: I have read Mr. Gray's paper through twice and think it is both comprehensive and exhaustive. Comprehensive, in that it covers almost all there is to be known about undersea mining; and exhaustive in that it deals with all the problems connected with such mining in our own and other countries. It is historical and tells the story of the past. The faithful and thorough work of former geologists is noted, as is also that of the practical miners of by-gone days. Their successes and their failures are pointed out—successes which have brought us to the present stage of progress, and failures which have brought us their penalties, just as the sins of the fathers are visited on the children.

It was not to be expected that the men of the past should have the vision we have today. As has been already said, one mile undersea seemed to mark the limit of their vision. But science came to their aid, as it will, no doubt, come to ours, and set the economic limit further off than we can predict. To me the paper is a very vivid picture of the past and the present, and it points to the future.

Unfortunately the warnings of such able men as Dr. Poole were unheeded by the governments and the mining men of the early days. He would have guided us along the way of science, but we preferred the sloughs and the by-ways. Science has ever endeavoured to make its paths clear, lighting them up with beacons of knowledge. Now we find that the way in some places has been missed and we must return to the trail again and follow it more closely.

Some of the outstanding problems of undersea mining referred to by Mr. Gray are ventilation, transmission of power, and methods of mining. His purpose seems to have been to cause mining men to concentrate on these problems, each having his own special problem to attack; and also to direct the energies of each in carrying to success his own part of the work. To help us he has gathered together a great many facts, much funded experience and information. And while he has drawn few conclusions the facts given are sufficient to build up our own hypotheses to help us unravel our own problems.

Many have searched for information on undersea mining and have been unable to find it, but here we have it compiled in one volume. If Mr. Gray's paper has started us to think along new lines he has done much. I know there are quite a number of mining men here today who think deeply of such problems as are set out, but the manager and underground manager who have to get the coal output have not time and are too tired to think of such matters. In the clear picture given them they have had a closer view which will help them. Mr. Gray's paper is a valuable contribution to the history of undersea mining, not only in Nova Scotia, but to mining literature the world over, and I congratulate him on the work he has done.

MR. H. C. M. GORDON: To the author of *Mining Coal Under the Sea in Nova Scotia* much credit is due. Mr. Gray in this, his most recent paper, has collected a wealth of fact which he has very ably presented. He has traced the history of the Nova Scotian coalfields from the time of their earliest workings to the present, and has pointed out the problems which will arise with future development and has sketched their probable solution. There was a great need for such a paper.

Of all these problems one of the most complex will be the ventilation of the submarine workings, at a distance exceeding four miles from the shore line, and under heavy overburden.

Let us consider the ventilation of the mines in the Sydney coalfield. At the present time it is a comparatively simple matter. Two or more intake airways, each approximately

65 square feet in area, are driven in the coal from the fan shafts to the farthest developments, and from these air splits are taken at required points. The roof in these intakes is supported by steel booms and the cross-cuts and other openings are sealed with 12-inch concrete stoppings. At present the loss through leakage seldom exceeds 10 per cent.

Subsidiary intakes are timbered either with props and caps or wooden straps, and the openings sealed with wooden stoppings faced with plaster. The whole of the abandoned workings are generally open to the return air and, in addition, travelling returns are maintained.

Ventilating pressures are low, only in one case exceeding 4-inch of water gauge, and the quantities of air circulated, while comparatively large in some cases, only average 91,000 cubic feet per minute for the seven operating submarine collieries. However, the seams, especially in the disturbed areas, are gassy and the ventilating current must be carried well into the working faces.

While the relative humidity in the working faces is high, generally about 90 per cent, the wet bulb temperatures are low, even at 1,300 ft. of cover being only 62°F.

Mr. Gray points out that the winning of the coal in the outlying submarine fields must be carried on by one of two general schemes: either by extension of the present and projected collieries to their economic limits, or by the sinking of deep shafts and cross drifting to the various seams.

Should the former method be followed, the first radical change in airway construction will be in the main seals or stoppings. The 12-inch concrete at present in use will be replaced by 10 ft. of puddled clay, supported on either side by 6 ft. of timber. This will be followed by developing the main airways as a longwall face, or a series of faces at least 300 ft. in total length. The goaf will be closely packed and the airways supported by circular girders and lagged with wood.

Nos. 12 and 14 collieries of the Dominion Coal Company will presently reach this stage at a cover of approximately 1,350 feet.

As an alternative to this packing, hydraulic stowing might be used. Probably the best method of doing this would be to develop the main sinking for the distance of one lift and then drive in to the projected airway and form a small settling lodgment. This work would be done as a closely packed longwall. The block to be flushed could then be won by a face driven up the dip at an angle of about 45 deg. to the line of development and the place flushed at every 10 ft. of advance. This diagonal line of face would provide easy drainage for the flushing water to the settling lodgment, from which it could be pumped when necessary. The permanent airway could either be formed in the centre of the flushed block as development advanced or could be driven at a later period through the flushing itself. In any case circular girdering with wood packing would be necessary. The cost of this method would be very high and it would only be used as a last resort.

In the lowest worked seam, or where only one seam was being worked, a third method of airway construction might be followed, that is, by driving drifts at a depth of about 50 feet below the seam in some suitable stratum. These drifts would either be lined with concrete or girdered, depending on the character of the stratum through which they were driven. This would be the best method of the three.

Subsidiary intakes would in all cases be developed as a closely packed longwall and would be supported by circular girders. A return airway would be maintained at the colliery barrier, and no more air than necessary allowed to return up the main sinking. In this way leakage would be reduced to a minimum.

As development progresses and resistances become greater, higher water gauges will be required to circulate sufficient quantities of air. However, as the mode of work will be entirely longwall, less air will be required than with the present pillar-and-room method. These two factors will balance one another to some extent for a considerable period but the more restricted return from the longwall workings will eventually increase the ventilating pressure necessary and the present intakes will have to be enlarged, probably to an area of 100 square feet. This latter will be the area of all girdered main airways. In addition, an exhaust fan will be required at the upcast to prevent excessive leakage as the mine resistances increase and will probably be installed when these approach the equivalent

of 10 inches of water gauge. It may even be necessary to install an auxiliary or booster fan underground at a distance of about three miles from the main intake fan. This, however, should only be done when all other methods of producing the necessary ventilating current have failed, because, in case of disaster, this installation would in all probability be destroyed, and the system of ventilation seriously impaired. Moreover, the heat of auto compression produced by this fan would be a serious objection to its being installed on the intake side of the workings. It would therefore seem that the proper position for an underground fan, bearing in mind the fact that the return is a restricted one, is in the main return airway and directly behind the workings. This would, of course, necessitate the taking of special precautions in the matter of ventilating the fan motor.

At a cover of 3,000 feet, a wet bulb temperature of 80 deg. will probably have been attained—the geothermic degree of the field is approximately 55 feet. It will then be necessary to resort to cooling the intake air by mechanical means. This will probably be done by an installation operated on the cold-storage-plant principle. Such a plant was erected in 1920 at the Morro Velho mine in Brazil and has been a successful installation. In this plant the intake air is passed through a series of air coolers of the Heenan type. These are short cylinders built up of steel plates in the form of a spiral. These coolers are slowly revolved and the lower segments passed through a tank of water which has been cooled by coils through which ammonia has been circulated. The air coming in contact with the steel is cooled to the required degree. The installation at Morro Velho is capable of cooling 80,000 cubic feet of air per minute from a wet bulb temperature of 72 deg. to one of 43.2 deg. It cost approximately \$400,000 and requires 700 e.h.p. to operate it at capacity. The cost of such an installation renders its use impossible in this field until the value of the coal has considerably increased.

If, on the other hand, the field is to be one from deep shafts and level cross-measure drifts cutting the various seams, the problem of ventilation becomes a serious one. Take for example an intake drift 200 square feet in area and a minimum quantity of air of 500,000 cubic feet per minute. The resistance

in the intake tunnel alone for a distance of 7 miles, i.e., to the Hub seam, and allowing each seam its proper proportion of the air, would be 20 inches of water gauge. A similar water gauge would be taken up in the return, and about 15 inches in the seams themselves. The mine water gauge would be, therefore, about 55 inches. Such a layout would require, therefore, at least two intake and two return drifts, and several auxiliary fans would be required in addition to the main forcing and exhaust fans. The method of ventilating the seams would be similar to those described above.

In addition to the excessive cost of building airways sufficient in number and in area to reduce the mine resistance to within reasonable limits, and to the doubtful practice of establishing a number of auxiliary fans, must be added the burden of installing a very large air-cooling plant. These together make the problem practically insurmountable to customary ventilating methods.

However, the problem may be solved by installing a large turbo compressor plant in place of the ordinary ventilating units and pumping air into the intake drift at a pressure of about 20 pounds per square inch. This drift, which would be carefully lined with concrete, need only be 12 ft. in diameter, and the return drift of a size sufficient for transportation purposes. At each seam a sufficient quantity of air would be released for ventilating purposes and delivered to the airways in that seam expanded to atmospheric pressure plus the remaining resistance of the mine, or the air could be carried through light steel piping into the workings and expanded there. In either case the expansion of the air would be sufficient to lower the temperature in the workings to such a degree that a high relative humidity would not be objectionable. This arrangement would, therefore, require no cooling plant and would be far less liable to break down than the ordinary ventilating units, while in case of disaster through explosion the ventilating system would probably remain intact.

MR. W. HERD: Mr. Gordon's contribution is very valuable, dealing, as it does, with what has to be done, according to his lights, in order to ventilate the deeper mines of the future. No doubt it will be a problem to keep the water gauge down. We may have to go to turbo-compressors and convey the air through comparatively large pipes.

MR. A. McEachern: The problem of deep mining and high temperatures is suggested in Mr. Gray's paper, and is outlined in further detail by Mr. Hay in a letter to me, in which he stated that:

"Undoubtedly one of the problems to be met with in prosecuting mining a long distance from the shore and at considerable depth, is that arising from the natural temperatures of the strata. Conditions as to temperature and humidity have a direct bearing on the accident rate in mines and also on the health of the workmen".

I think it would be quite in order at this stage to remind Mr. Gray, that he, more than any other mining man in Canada, helped to arouse the public mind to the greater use of coal and its wider application in varied forms. Even his present paper contains reference of this kind. This naturally excites the mind to the idea of higher prices for coal. If this is brought about in the course of time, will it not have the effect of delaying the evil day of 'economic limit' and push it further back, bringing it into closer relationship with the engineering and other problems forecast by Mr. Gray. The growing scarcity of other fuels, for which coal must of necessity become a substitute, will, no doubt, naturally affect the price of coal and send it up. This is but a passing reference to cast a beam of light on the darkest side of the subject under discussion.

Most of the statements I shall make are based on the observations and experiments of scientific men who had the opportunity to study this important question of deep mining at close range over a number of years. The deeper coal mines of Nova Scotia are among that class of collieries whose atmospheric conditions are affected by depth, and it seems necessary that we should know something of the problems which lie before us and how these are met and dealt with in other places.

The purpose of my remarks is merely to point out what others have done under conditions which mining in Nova Scotia must inevitably meet in the not far off future. In doing this, I shall find it necessary to quote largely from reports prepared, submitted and published by British and other scientists.

There are three causes for the rise of temperature in underground workings. The first is the heating due to compression of the air as it descends shafts or inclines; the second

is conduction of heat from the surrounding strata; and the third is formation of heat in the oxidation of coal or other minerals. The increase in the moisture contained in the air is due to evaporation of the water which percolates through the wall of shafts and roads, or is present in freshly exposed coal or other material. The heat and moisture given off from men, horses, and lamps is scarcely appreciable in a coal mine.

Heating by compression cannot be obviated, except at the expense of evaporation; but it can be, and to a large extent is, averaged over the year in the actual workings of a mine, the walls of the shaft and roads giving off more or less heat to the air according as the external temperature is lower or higher. In exceptionally warm weather the walls may even take up heat from the air.

Heating by conduction from the surrounding strata is reduced to a minimum in shafts and roads when the volume of air passing is sufficiently large. This is because the flow of heat through the zone of cooled rock which comes to be formed in course of time round a shaft or airway is comparatively slow. If the air flow is large, the flow of heat from the strata is insufficient to warm the air more than very slightly; but if the air is flowing very slowly, the heat flows sufficient to warm it to practically the temperature of the strata.

It should be noted here that there is a wide variation in strata-temperature in different countries, more especially between that of Britain and South Africa. Different beds of strata have different conductivities. Hard and completely compact rocks, such as igneous rocks, have a high conductivity, whereas strata of sedimentary origin have a comparatively low conductivity. Coal-measure shale is a better conductor of heat than the coal itself, being in the ratio of three to one. The nature of strata, as to whether it is wet or dry, also has an important bearing on its heat conductivity. A wet sand-stone is a better conductor of heat than a dry sandstone, approximately in the ratio of three to one. Strata temperature varies with an undulating surface. It makes a difference if a bore-hole is under a mountain or in a valley.

In Britain the rate of increase of the natural rock temperature with depth is always much greater than the increase of air temperature from compression; and at collieries where experiments are carried on the rate of increase in rock temperature was one degree Fahr. for about every 60 or 65 feet, whereas the air temperature increase from compression was only one degree Fahr. in 182 feet.

In the Witwatersrand district, South Africa, however, the rock temperature increase is only about one degree Fahr. in 250 feet, so that if a deep mine were dry, abundant ventilation would warm it, in contrast to the cooling effect in Britain.

The effect of high temperature underground is shown by experiments carried out in the Pendleton colliery, England. The work was undertaken by the Hot and Deep Mines' Committee. Professor K. Neville Moss presented a statement of the results, which were, in part, as follows:

The first section of the work dealt with the food and drink requirements of miners in relation to underground

temperature.

After giving a short account of the scientific principles of converting food as the fuel of the human machine into energy in the form of calorie values, he passed on to the regulation of body temperature during work in hot air. He went on to state that the control of body temperature is a function of the nervous system which accomplishes the work either by chemical or by physical means. Leaving out the chemical control, as it did not apply in his report, he dealt with the physical means of control as being of great importance to the miner. The rate at which blood is supplied to the skin influences the heat-loss by radiation and conduction, as, for example, during work in a cold air current; but when a miner is at work in a temperature approximating, or even exceeding, that of the normal body temperature (98 degrees Fahr.), the normal conditions can be obtained only by evaporation of water from the skin and lungs. If, however, an increase in body temperature during work takes place, it can only do so by a disturbed relationship between an abnormally high heat production and a heat eliminaton which cannot keep pace with it. The quantity of liquid drunk per day is shown to increase rapidly with the increase of underground air temperature above 70 degrees Fahr.

The following table has been compiled from experiments carried out at Hamstead and Pendleton collieries to determine the body loss in weight, due to perspiration and breathing, of

the thirteen miners undergoing the test:

BODY LOSS IN WEIGHT DUE TO PERSPIRATION AND BREATHING

Name of colliery	Under- ground temper- ature, in deg. F.	Case No.	No. of experiments on each	Maximum loss per shift— 5 hours (in lb.)	Minimum loss per shift— 5¾ hours (in lb.)	sweating per hour
						(in lb.)
Pendleton	98–100	1	2	18.56	15.25	3.175
	deg.F.	2	2	18.75	18.00	3.145
	dry,	3	2	16.12	15.44	2.695
	85°F.	4	2	17.12	11.81	2.67
	wet.	5	2	13.68	12.5	2.475
		6	2	12.44	12.12	2.28
		7	2	12.68	10.81	2.205
		8	2	12.12	11.31	2.185
		9	2	12.94	10.44	2.168
		10	2	13.75	9.68	2.16
		11	2	13.25	11.62	2.135
		12	2	10.56	10.12	1.93
		13	2	9.37	9.31	1.76
Hamstead	82°F.	A.C.	1	10.19		1.85
	dry,	D.T.	28	11.81	8.25	1.70
	77°F.	RS.	5	9.25	7.12	1.42
	wet.	C.P.	3	9.06	5.94	1.33
		W.D.G.	1	6.62		1.20
		W.T.	4	7.25	5.19	1.17
		K.N.M.	1	4.69		0.85

The tests carried out on the thirteen Pendleton miners showed that the maximum loss from skin and lungs amounted to 18.56 lb. for five hours' work, and the minimum loss to 9.375 lb. for 5.75 hours' work; or, in other words, to 3.7 and 1.66 lb. per hour, respectively. The average volume of urine passed per man was 155 c.c. per shift, which is only half of that of a man working under normal conditions of temperature. The average amount of water drunk per shift was $7\frac{1}{2}$ pounds. Thus, the water drunk did not compensate for the loss by skin and lungs, the bulk of which, of course, was lost by the skin.

As the sweat contains a very appreciable amount of chloride, it is evident that the miners were losing much chloride during this work.

It has often been assumed that the chloride in sweat is almost entirely sodium chloride, or common salt, since sodium predominates very greatly over potassium and other basic substances in the blood plasma. An analysis made by Mr. J. Ivon Graham in connection with one of these experiments showed, however, that 59.3 per cent of the chloride was sodium chloride and the remaining 40.7 per cent potassium chloride.

Before commencing an experiment, the subject is washed down very thoroughly with hot water, dried and then weighed after first passing urine. The work is performed in thin cotton 'shorts' which have been washed in distilled water, dried and weighed before use. Exact weights are taken of food eaten, water drunk, and urine passed during any experiment. The volume of air expired is measured by an airmeter during the performance of a given amount of work, and also during rest. This information enables one to estimate approximately the moisture by respiration and obtain the loss due to perspiration only. The loss of perspiration in pounds per hour, as shown in the above table, is, therefore, actual; the respiration loss having been deducted. The latter loss, was, however, very small.

From these experiments the following conclusions result:

- (1) For fixed conditions of temperature and humidity an increase in the work output is accompanied by an increase in the sweat loss, and apparently also of its sodium chloride content. Also, if the work output be maintained constant, an increase in the loss of sweat and its sodium chloride content accompanies an increase in temperature. Where there is much sweating, the loss of chloride from the body is very large and may be enormously greater than the loss by the urine. This great loss of chloride by sweating is evidently related to the extra quantity of sodium and potassium salts in the diet of miners working in hot mines.
- (2) There is a marked difference in the amount of sweating between the men acclimatised to hard manual work under high temperature conditions, and the men who take

just sufficient exercise to keep fit. It is seen that for the same work output under nearly equal conditions of temperature and humidity, the miner loses more than twice as much weight by sweating as do those not acclimatised. When, in a further experiment, a miner was 'pressed' by an increase in the dry and wet bulb temperature, he lost 5.8 lb. per hour by sweating, which is a remarkable figure; and yet, in a more recent series of experiments conducted upon another Pendleton miner, the maximum loss per hour by sweating amounted to 6.55 lb.

While many miners work for years in deep hot mines without any seeming harmful effect, it has been observed that where the temperature varies from 98 degrees to 102 degrees Fahr. dry bulb, and 83 to 87 degrees wet bulb, miners' cramp and heat stroke take place. There seems to be no doubt, however, that even where cases of severe cramp do not occur the working capacity of the miner is reduced by the same cause as leads finally to the attack of cramp. In severe cases of cramp men are carried out of the mine. Such cases, however, are rare, as only nine were known to have occurred in two years in a large deep mining district.

Cramp is due to the following causes:

- (1) High air temperature;
- (2) Excessive drinking of water due to temperature; and
- (3) Continued hard work.

Men are generally affected by cramp during the latter half of the shift, and always in the muscles being strained at the time. Evidence collected by Dr. J. S. Haldane from men in the mine led him to suggest that cramp may depend upon excessive loss of chloride by continual sweating. Experiments carried on by other eminent men coincide with the opinion of Dr. Haldane. Dr. Priestly states that when excess of water is voluntarily drunk, water poisoning takes place followed by miners' cramp, and with it all the symptoms of fatigue that accompany cramp. To prevent water poisoning and cramps, experiments were carried out by placing salt in the miner's drink to balance the loss by sweating. Among the number of men selected were sufferers and non-sufferers by cramp, and men of good and poor physique. The results varied according to their likes and dislikes for salt. Later

experiments pointed to the efficacy of using a mixture of potassium chloride and sodium chloride, in the proportion of 40-60, in the drinking water. Many miners put cream-of-tartar into the water carried by them to the mine, as a cramp preventive.

The experiments conducted showed that when salt was taken in the drinking water during work periods the fatiguing effect was less felt than when salt-free water was used.

The experiments on the thirteen miners indicate that the human body can acclimatise itself to almost any condition of temperature likely to be met with at depths down to 5,000 ft., and yet, if a man unfamiliar with mine work and high air-temperature were to attempt to do much hard muscular work in a deep hot mine, he would, in all probability, have a heat stroke.

Heat strokes occur in the deep mines in South Africa. On January 13th of this year a most interesting paper was read before the South African Institute of Engineers on this subject by two doctors who had made an inquiry into four cases of sudden death in the Village Deep mine, which is more than 7,000 ft. deep. They stated that the first signal is the skin going dry. As long as a man is sweating freely he is fairly safe, and it is the exhaustion of sweating that precedes heat collapse. If a person working in a hot place is noticed to have a dry skin he should be watched carefully, or sent out. If collapse occurs, artificial sweating is brought about by sprinkling the body with water and blowing air over it. It is important then to keep the skin moist until the person is handed over to the doctor. Heat stroke is cardiac failure the heart has been doing double work in its attempt to keep a large volume of blood flowing rapidly through the superficial vessels while maintaining an adequate supply to the working muscles.

Because of the reduction of heat by the cooling power of the air and evaporation of water in shafts and roadways, temperature due to compression of air and strata conductivity is usually low. This is shown by the following illustration taken from one of many similar experiments made in deep mines:

Pendleton colliery shaft is 3,526 feet deep and 8 feet in diameter. With a surface temperature of 50 degrees and air travelling at the rate of 40,000 cubic feet per minute, the temperature at the pit bottom was only 61 degrees. At the end of intake, due to compression and strata heat, the temperature was 76 degrees, while due to both of these causes and that of oxidation, which is always higher near the working faces, the temperature was 104 degrees dry bulb and 83 degrees wet bulb, the relative humidity being 41. The low rise in the shaft was due to the cooling power of the air on the strata and to evaporation of the water from the shaft walls.

Heating by oxidation can evidently be controlled by abundant ventilation. Mr. F. Jensen states that the heat of oxidation is greater in proportion as coal is scattered about—as in curves, landings, and chutes. This, as well as long and close contact with the air, as on the conveyors, and the high temperatures from loose goaf stowage, all exert a heating influence equivalent to nearly 12 degrees. Of this, 4.1 degrees arise in the intake airways and could be entirely prevented by ceasing to haul coal through the intake current; but in the workings the heat due to oxidation could hardly be reduced.

Increase in moisture content of the air is a factor which can usually be controlled as effectively by ventilation, until the actual working face is reached, as can increase of temperature. But with abundant ventilation, the cooling effect of evaporation from wet roads would eventually keep the temperature along these roads low, and thus indirectly control the amount of moisture in the air and the wet-bulb temperature. There is a very rapid rise in moisture content of the air as it passes along a working place over fresh and relatively moist coal.

Experiments were carried on to find the cooling power of air. Tests were made with the air still and also moving at varying velocities. It was found that the rate of cooling rose with the velocity, and that this rate was proportional to the square root of the velocity over the range of velocity observed.

The fact that the cooling power of the air is proportional to the square root of the velocity has interesting practical results. It means that a moderate ventilation has a great effect on cooling power. Thus, the cooling power of the air at a velocity of 120 feet per minute is double that of stagnant air. In order to secure the same effect by decreasing the temperature, a reduction of 10 to 20 degrees Fahr. would have to be made at high wet bulbs. Further, it means that increasing the ventilation from a stagnant condition to a velocity of 100 feet a minute has a much greater effect on the cooling power than increasing it from 200 feet a minute to 300 feet. Hence, it is worth while taking trouble to secure even a slight movement of the air in places that are very hot, as even a small movement has a marked effect on the cooling.

Another important fact that must be taken into account is that the miner is moving to a greater or lesser extent all the time he is working. Where the ventilation is weak this must have a considerable effect in increasing the cooling power of the air on the miner. In the case of a man walking against an air current moving at a fairly high velocity, the cooling power of the air increases at least fifty per cent. A man pushing a mine car against the ventilation will, of course, experience much the same benefit.

In the submarine workings of Lavant mine, in Cornwall, the wet-bulb temperature was over 90 degrees. The men worked for a short time and then cooled themselves at jets of compressed air and the work went on, though, of course, very slowly.

It appeared that during rest in perfectly still air and with clothing removed to the waist, a wet-bulb temperature of 88 degrees was the limit which could be withstood without a progressive rise of body temperature. During moderate work the limit went down to about 78 degrees wet bulb, but in an air current of about 170 feet per minute, the limit during rest went up to 94 degrees and during moderate work to about 85 degrees.

With the dry bulb at 104.9 degrees and the wet bulb 87.8 and air velocity 300 feet per minute, cooling power of air was 9.4, and the relative humidity 39 per cent. With the dry-bulb temperature at 110 degrees and wet bulb 86.6 degrees, and air velocity 1,337 feet per minute, the cooling power

was 17.4 and the relative humidity 40, or four and one-half times the velocity for double the cooling power. This is further illustrated by the following table:

1	Temperature	Cooling power		
Dry bulb, deg. Fahr.	Wet bulb, deg. Fahr.	Velocity, in ft. per min.	Observed	Calculated
111.9 96.1 110.6 111.9 112.0	87.8 78.1 92.0 87.5 87.4	300 210 340 494 672	10.4 18.0 5.5 11.5 13.1	9.4 16.8 5.3 11.0 12.2 17.4
112.0 109.6 86.6		672 1,337	13.1 16.3	

Temperature and humidity, according to experience, are the two most important atmospheric conditions that affect the health and efficiency of people in their living and working places.

Humidity increases the discomfort and ill effects of both high and low temperature. At 32 degrees, the effective temperature line coincides with the dry-bulb temperature line. In the comfort zone, comfort depends equally upon wet-bulb and dry-bulb temperatures. Here, the optimum temperature at rest is around 66 degrees Fahr., and at work the temperature is 59.5 degrees; 43 degrees has been pronounced as too cold, with or without air movement, regardless of humidity.

Air movement decreases the discomfort of high temperatures below 98 degrees Fahr. Moving saturated air above 98 degrees Fahr. has been found to be of no benefit and is even disadvantageous.

Death generally takes place when the body temperature is raised to between 109 and 113 degrees Fahr.

It has been stated that the minimum accident frequency varies from 75 to 89 degrees Fahr. At temperatures above 75 degrees it was 35 per cent greater than at 65 to 69 degrees. This difference has been attributed to the effect of greater exertion at higher temperatures.

At Pendleton colliery, with deep workings extending to 3,700 feet, the wet-bulb temperature at the longwall faces was as high as 83 degrees. The miners worked stark naked and got as much coal as usual. They admitted that they could not do as much physical work as in cooler places. Up to 70 degrees wet-bulb temperature, there is comfort at work. Above this, discomfort begins, but there is a wide margin before the limit is reached.

Investigations by Mr. D. Harrington, of the United States Bureau of Mines, have led to the conclusion that high humidity of underground air is not disadvantageous to health or working efficiency until the temperature is over 70 degrees Fahr. If the air is stagnant and the humidity is over 85 per cent, high temperatures are very harmful to health and working efficiency, but if the air has a velocity of a few hundred feet or more per minute, temperatures above 90 degrees Fahr. can be endured and are apparently not unhealthful, even if the humidity is high. The same authority has found that at least 90 per cent of underground working faces have a relative humidity above 85 per cent, irrespective of the region in which the mine is located, the time of the day, or the season of the year.

Dr. Haldane states that abundant ventilation is certainly the readiest means of meeting all the immediately prospective temperature difficulties in deep mining in Britain or elsewhere. But in a recently published report on deep mining in South Africa, it was stated that, at a depth of 7,032 feet, ventilation presented a very serious problem. In these mines, ice cooled air and water have been tried with good effect, but this was confined to local districts—economy of operation prohibiting a general system of ice cooling.

Dr. Leonard Hill, applying the air-cooling principle to factories, states that, if the right cooling powers could be had for different kinds of work, there would be greater efficiency. He believed that they could remove an enormous amount of industrial unrest by banishing the stuffy atmosphere of factories, which depresses the vitality and lessens the food-burning power of the body, and causes the heart to be taxed to cool the blood, by sending it to the skin where sweating takes place.

To sum up briefly, we find, that:

- (1) Deep mines have high temperatures.
- (2) These temperatures vary as the depth and nature of the strata.
- (3) High temperatures affect the working capacity and health of the miner.
- (4) Good ventilation is the remedy in mines of high temperature. This may be supplemented by aircooling systems.
- (5) Evaporation of moisture, velocity of air, compressedair jets, and ice in the air and in water sprays of the mine, reduce temperature.
- (6) Where heat produces great loss of sweat, the use of salt in drinking water reduces the effect of fatigue.

From the above it will be seen that large airways are most necessary in deep mines with extensive workings.

MR. HERD: We are deeply indebted to Mr. McEachern for having prepared an elaborate discussion on Mr. Gray's paper. He has treated the matter of future undersea operations from the physiological standpoint and has shown that a good day's work cannot be done in deep mines without the aid of better ventilation. I have heard that men under test lost two gallons of perspiration in deep English mines. Mr. McEachern says about nineteen pounds, which is practically the same. Those not accustomed to vigorous manual labour in deep mines find conditions most onerous, and during a visit I made to one of these collieries I considered that I was doing a good day's work standing still. Dr. Haldane, who advocated salt, got his idea from the miners' wives, who told him their men preferred salt ham when they returned from the pit. proper ventilation of these mines is looked upon in England as a very important subject.

MR. James McMahon: The present and future method of mining for coal in submarine areas, as indicated in Mr. Gray's paper, must cause every department that enters into the underground operations to consider seriously what means are to be taken to meet the problems that will arise from the abnormal conditions that will prevail as the working faces recede farther and farther from the shore line.

After reading Mr. Gray's paper carefully, I tried to picture the conditions that would have to be met in the rescue and recovery of a mine, in the event of disaster arising from explosion or fire, or both together.

Our present methods of attack are based on the work

that can be done by

(1) Men working bare-faced and following regular mining practice of making the roads safe for travel and carrying the air in with them as they advance;

(2) Men equipped with smoke helmets or equalizers, always working from 150 feet to 200 feet in advance of the fresh air base, and followed closely by the bare-faced workers; and

(3) Men equipped with self-contained breathing apparatus, who would be able to advance to much greater

distances from the base of operations.

Now, while there is sure to be progress in means placed at our disposal for rescue and recovery work as the years pass, and the organization and methods of attack are improved and consolidated from experiences gained, I feel safe in saying the only chance the man left alive in-bye at the time of disaster will have to escape will be by means of provisions that have been made in advance and located close to the workings in active operation.

What form may the aids of escape take that would appear to offer a probable solution of the conditions contingent on a disaster?

The utilization of old workings has been advocated by the U. S. Bureau of Mines, which arises from the fact that, on several occasions, men have retired to old workings, built barriers of brattice and dirt, etc., and have sat quietly behind these till help has reached them, instead of facing the main roads out-bye with the almost certain result of being asphyxiated by the after-damp, etc.

The Committee on Rescue and Recovery Standards appointed at the St. Louis Convention in 1921 reported about end of 1923, and advocated the provision of explosion-proof chambers as refuge places to which men might make their way and remain in comparative safety till help came from the surface.

And lastly, I think the time is opportune to arrange for strictly isolated roads through the worked-out areas, extending from the pit bottom to the last level working, such roads to be thoroughly dammed-off right through to the first working level and from there provided with an entrance from an offset road, closed by sets of explosion doors at each level entrance; these doors to be removed and the entrance dammed as the section is finished. These roads would be of greater value if a line of compressed air piping were carried through with valves at regular intervals to enable the roadways to be flooded with compressed air from the surface in case of need. A telephone system also, should be carried through the escape roads.

The values of the three methods of escape for those trapped below are very unequal.

To take the first one: The utilization of old workings would be the cause of so much doubt to those attempting recovery that they would hardly know which way to turn unless some plan were definitely outlined as to the parts available for men to retire to for safety, and a limit of the area they should traverse properly understood and adhered to, a condition difficult to guarantee.

The explosion-proof refuges advocated by the St. Louis committee have been assumed to be used as first-aid chambers, so becoming known to the workmen generally, and so likely to be the places they would make for if the necessity arose. They are also indicated as being bases from which rescue and recovery operations would function. There will be one great difficulty that must be overcome to prevent such places becoming death-traps, instead of refuges—the provision of oxygen in sufficient quantity, and the removal of the carbon dioxide from the air, both absolutely necessary conditions to support life, even assuming that no after-damp could find its way into the refuge.

The last idea of a real escape-way—isolated, with its explosion doors outside, and an air-tight stopping with a sliding regulation door to permit entrance, and with the excess pressure of air from the compressed-air line to cause the flow to be outwards and thus keep the air breathable—

would enable the men trapped gradually to make their way outwards, and those attempting rescue and recovery work to get quickly to their assistance.

Roads of this character would be in the truest sense of the word escape ways, unlike the so-called escape ways of the present, which are part of the intake or return system of the mines where they are provided.

Dr. Garforth, in his Rescue and Recovery Work in Mines, advocated that all men be provided with a self-rescue apparatus of the half-hour or one-hour type. This would enable each man uninjured to make his way to the escape road or refuge even though he had to travel through after-damp. No doubt, in the mining of the future, such apparatus will be provided for each man, who will be taught how to use it, keep it in good condition, and be called on to produce it periodically for inspection, so that it may be in satisfactory condition when needed. The value of the escape roads will be doubtful, unless every man employed below is thoroughly acquainted with their location.

To insure this, I would insist that each man, without exception, be required to travel either in or out at least once every three months; and about this same time he should be made to produce his self-rescue apparatus for inspection.

The scheme outlined by Mr. Gray for the working of seams by means of a deep shaft and cross-measure tunnels will greatly complicate matters in case of a disaster, unless each seam and its workings can be isolated completely from every other seam, in which case, to enable advantage to be taken of such isolated areas, a drop staple pit could be driven to connect the escape roads in each seam, and so enhance the chance of escape from the mine involved. What will have to be faced under the new conditions is in the nature of a sealed book, for, in nearly all cases where many seams are won from one pair of shafts by cross-measure drifts, there is generally an opening to the shafts independent of the drift, thus giving more than one way in or out. In the scheme advocated by Mr. Gray, entrance and exit will be at the place where the tunnel intersects the seam, unless by some such means as I have outlined above—namely, drop staple pits from escape road in one seam to escape road in the one above or below. In all cases, full use will have to be made of protected telephones and lines; water for drinking stored in the refuges; enough food to last a moderate siege; spare self-rescue apparatus; and connection with the compressed-air lines; or, alternatively, a supply of compressed oxygen and some provision for regenerating the air, also an adequate supply of first-aid material and stretchers, blankets, etc.

Our responsibility to the human working below is the first call for safety measures, but an explosion or some other agency may cause fire—a persistent, pernicious foe to overcome. Here again the escape roads would be our salvation, for the compressed-air lines would be available for converting to the water supply and connection rapidly made with the seat of the fire.

One thing I would like again to stress—the necessity of every man and boy being made familiar with the refuge he should make for in case of need. Also, if escape ways are established, see that they are travelled by every one in the mine once in three months. Examiners, shot-firers, and overmen would be the better if shifted around the various districts of the mine or made to travel the districts foreign to their own periodically; otherwise, they will be useless to act as leaders to those under them.

I do not think that 5 per cent of the men employed in No. 1-B and No. 2 mines could find their way unaided through the escape-way; nor do I except the officials above-mentioned in this respect. Greater use should be made of prominent notice boards and sign posts giving direction of travel, and placed in such manner that he who runs may, or rather must, see and read.

There is bound to be progress in the organization and methods of recovery and rescue work in the next generation or two, perhaps much greater than anything accomplished in the past 25 years; but even if present methods are greatly improved, there is sure to be an appreciable period elapse between the time of occurrence of a disaster and the arrival of rescue men in-bye to aid those trapped. It was to shorten this period as much as possible and to enable the trapped men to have the greatest chance of self-rescue that I have tried to sketch out my idea of the escape roads. They may not

appeal to the mining engineer for many reasons: Original cost of construction, cost of maintenance, and the bare chance that they will ever be required in his time. But in undersea mining the unexpected happens, as in other forms of the mining game, and he is the best mining engineer who is ready for it.

To that end a full and complete organization should be established on the surface to take from the shoulders of the manager all unnecessary tasks. One man should be detailed to order our rescue crews, firemen, doctors, police, and all necessary aid; another to be responsible for suitable feeding and sleeping arrangements for those working; another to arrange a temporary hospital and mortuary; another to be absolutely in charge of the entrance to mine to see that all men are carefully checked in and out, and only those authorized permitted to go below; another to arrange brattice, timber and supplies. Many other arrangements might be suggested, but these may be taken as typical.

Also, some proper division of duties to the underground officials should be made in advance. Such offices to be allocated to two men, who would operate on different shifts, so that one would always be available in case of need; a third official thoroughly acquainted with old and new workings, should be appointed to take full charge in case the manager and underground manager are not available, and, as nearly as possible, the underground force taught the most feasible way of getting to safety and of helping save others and the mine.

MR. HERD: We are obliged to Mr. McMahon for his remarks, as he is well acquainted with rescue work in the mines. I cannot, however, agree with him in the case of escape ways.

MR. J. E. McLurg: Mr. Gray, in taking the chair this morning, referred to the education of our officials, mentioning the lack of facilities for the education of young men who intend to make coal mining their life work. It is difficult to get mine managers and other officials to carry on operations; that is, men with sufficient general knowledge to efficiently fill these positions. A copy of Mr. Gray's paper should be in the hands of every coal mine official in the Province. It also should be distributed to men working at the face who have ambitions to become officials

While I am stressing Mr. Gray's remarks on this point, I am not minimizing the other points he brought up. Mr. Gray's paper is a distinct contribution to the industry as a whole. I would like to speak on the financial end of coal mining. From an investment point of view, coal mining is the most hazardous in the world. Mining has been carried on in Cape Breton for one hundred years, and we seem to lose sight of the special hazards of coal mining as compared with other mining, although there is not much difference, in the mind of the general public, between coal and metalliferous mines. The Dominion Coal Company is the largest coal mining organization in Canada, with assets of thirty-two million dollars. There have been issued five million five per cent bonds, three million seven per cent preferred stock, and fifteen million common stock. holders of preferred stock have received no return on their investment since March, 1924. There is \$21 dividend accrued on each share, which they will get some day, if the industry succeeds. Many years ago, when Mr. James Ross sold common at \$95 a share, the industry was much more prosperous than it is today. I make these remarks for the reason that we all earn our living out of coal mining and our success depends on the success of the industry.

Mr. Gray has projected the future mine. This is not new, as it has been discussed by the 'Besco' executive for the last two years. A mine such as Mr. Gray refers to will cost between eight and ten million dollars.

Mr. Gordon, in his remarks, referred to an installation, at a mine in Brazil, for the purpose of cooling the air, which cost approximately \$400,000. Such an installation is impossible in Nova Scotia until coal reaches a higher commercial value. A discussion of this matter is all right, but in our discussions we must not be too theoretical, and we should give some thought as to who is going to put up the money for these improvements, and how much return the investment will pay.

MR. HERD: We are indebted to Mr. McLurg for bringing home to us the financial side of the coal mining industry. Coal mining companies are not philanthropic institutions and coal must be produced at a figure that people will pay. Coal

mining under the sea will, in the future, take place at remote distances, and we are here to discuss what methods seem feasible to overcome the increasing cost due to depth and distance.

COLONEL THOS. CANTLEY, M.P.: It is now three and thirty years, or perhaps a little more, since I first attended a meeting of the Mining Society of Nova Scotia, and I am delighted to be here today. But what a change in the personnel of the Society, especially the active, controlling, and participating members, the years have brought.

I have always been interested in the work of the Society, particularly in the discussion of the really able and instructive

papers prepared for and read at these annual meetings.

The paper just presented by our new President, Mr. Gray, is a classic and will be quoted from and referred to in after years as an authoritative, historical document relative to submarine coal mining, not only in Nova Scotia, but also in Great Britain and foreign countries.

Mr. Gray referred to the General Mining Association as beginning work in 1827. My impression is that 1826 is the date of their advent here. This I do know, that their Charter, a perpetual one, was dated 1826, and that the Association was in active operation at that time. However, Mr. Gray can readily determine these dates, and no doubt will now do so.

The first submarine mining in Nova Scotia was not done in Cape Breton but in Pictou county, when the G.M.A. worked the thick seam at the Albion mines and carried their workings southeasterly across and under the two branches of the East river (one of the greatest rivers in Canada) as early as 1832. Some trouble was experienced due to overhead bodies of water.

I would have liked Mr. Gray to have referred at greater length and more fully to the work of Richard Brown, and the great, important, and unique place he occupied in Nova Scotia's coal mining history.

No man of all those who landed on our shores did so much of a permanent character as regards the locating, tracing, proving, and correctly estimating the relative value of the various seams of coal in this Province as did that gentleman. Not one coal seam has been discovered, not an original opening

made, not a field explored, that had not been referred to by Richard Brown. I submit, therefore, that a classic such as Mr. Gray's paper is, should have dealt generously with the 'Great Old Man' of our coal mining fraternity.

I sometimes talk with D. H. McLean, resident superintendent of the Acadia collieries, about the problems of the future, and according to him there will be no difficulty in getting men to the face, or getting the coal away. He declares that the men can be sent in by an apparatus similar to that used as cash carriers in large department stores, and the coal can be taken out by crushing it and blowing it along by the out-take air. Getting men down the shaft will be just as simple. All that will be required is a certain amount of faith.

I have contributed nothing from a practical point of view, but nevertheless I am very much interested in the discussion.

MR. HERD: There is a section in Mr. Gray's paper dealing with the deposition of the Sydney coal field. It is highly theoretical but interesting, and a discussion on this phase may tend to bring out something of value. As Dr. Bell is the most qualified to speak on this, I would ask him for a few remarks.

DR. W. A. Bell: Mr. Gray's paper is certainly the most comprehensive summary of the facts pertaining to the Sydney coalfield that has yet been written. In that alone it is a scholarly effort written in simple, direct English, intelligible to all readers. But Mr. Gray goes much further and builds upon his facts as a foundation a broad practical tower of observation from which he views the vital problem of future coal exploration in the Province, a problem upon the solution of which much of the future welfare of the Province will rest.

To this end Mr. Gray makes appeal for the closest cooperation between the mine management, the chemist, the geologist, and the Provincial and Federal Governments. His viewpoint, too, is sufficiently broad to appreciate the economic value of what is sometimes designated pure scientific research. It is well that this is so, for until this attitude becomes more general among industrial managements and the public at large, it is useless to hope that the governments concerned will feel in a position to vote the necessary appropriations. I shall confine my discussion of Mr. Gray's paper to the subject with which I am most acquainted, that of the sedimentation and conditions of deposition of the Carboniferous deposits, of which the coal seams are but one expression. It is reassuring to find Mr. Gray appreciative of even the economic value of this phase of research. I shall crave your permission, therefore, to digress somewhat and outline the salient features of Carboniferous sedimentation in the Province.

In the beginning we have the prelude of Carboniferous conditions in the great mountain-making disturbance of middle Devonian times, called the Acadian or Schickshockian disturbance. As a result of this disturbance, mountains were upheaved in the Gold Belt of southern Nova Scotia and inland, then existing to the south, as well as in central New Brunswick and the Gaspé peninsula. Sediments ante-dating this disturbance were in most instances strongly folded and were intruded by granitic masses, of which the magmas were accompanied by gold-bearing solutions.

In upper Devonian times there was a long interval of erosion, whereby the mountains wore down to uplands of only moderate relief; and at the beginning of lower Carboniferous time we have a broad plain or valley of river deposition lying between the subdued uplands on the south and those in central New Brunswick. On this plain, lower Horton freshwater deposits were laid down. Environment conditions at times approached very close to those of coal formation, and in a few localities these coals did form, but nowhere of workable extent. The broad lowland valley of deposition was progressively subsiding, so that several thousand feet of river and lake deposits were preserved from erosion. A subsiding basin of this character is called by geologists a geosyncline. At no time was the subsidence of the plain of deposition sufficient to flood the Horton plain with the sea. The subsidence, however, was not constant, but was intermittent in The long-continued supply of sediment from the bordering uplands indicates that these were subject to a complementary intermittent elevation. These conditions of sedimentation of the Horton plain are referred to because they paralleled very closely those of later coal-forming epochs.

About the middle of Horton time the upland areas, which may be termed the positive masses in opposition to the negative geosynclines, underwent an uplift more severe in its effects on the sedimentation, and at this time what appears to be the locus of maximum sedimentation in the subsiding valley was upfolded in a narrow linear ridge to form the Cobequid mountain range. Subsequently, this Cobequid ridge was positive in character and it influenced greatly sedimentation in the river valleys on either side of it. As a result of the Cobequid uplift, we have a Cumberland geosynclinal valley in the north and a Riversdale geosynclinal valley in the south. The Cobequid positive ridge runs about east and west. To the eastward of it, a second positive area running northeasterly was established. represented at the present day by the upland south of the Thorburn coalfield, and by its continuation in the Antigonish hills. These two positive masses were separated by a structural gap, which may be termed the Stellarton gap, which was at times positive and at other times negative in character, so that the geological history of the Pictou coalfield is the most complicated of any in the Province. The river system that drained the Riversdale geosyncline seemingly found an outlet at times through this gap. Other positive areas were seemingly established or rejuvenated by reason of the Horton disturbance, including such positive areas as the Caledonian upland of southern New Brunswick, and the Craignish and other uplands in Cape Breton. The diversity of surface relief thus established is reflected in the upper Horton sediments by the presence of thick alluvial fans and coarse conglomerates or, in general, by deposits of intermontane character. The climate, too, had changed from pluvial, perhaps cool, conditions in lower Horton time to warm semi-arid conditions in upper Horton time.

Towards the close of the upper Horton, a time of greater crustal stability permitted of erosion of the uplands and infilling of the valleys until the surface relief was again very moderate and approaching that of a lowland plain. A general seaward or eastwardly tilting of the whole region was then effected, whereby the Windsor sea found entry, first up the broad mature valleys and later by transgressive overlap over the low uplands. The waters were shallow, the climate still warm and semi-arid; the crust still unstable, with intermittent

movements along the old positive and negative axes. Consequently saline deposits are an important feature of the Windsor marine sediments.

Crustal instability within the Windsor epoch, as within the Horton, came to a peak. The Cobequid ridge underwent an accentuated uplift, and warping movements of a broader character shut out the sea entirely from the western part of the Cumberland geosyncline, and throughout the remainder of Windsor time this part of the Cumberland valley was the seat of fresh-water deposits, mainly coarse alluvium from the rejuvenated uplands. Marine conditions prevailed south of the Cobequid and in Cape Breton to nearly the close of the lower Carboniferous period.

Regional uplift then thrust out the sea completely, so that no deposits of marine origin are known from the upper Carboniferous onwards.

Upper Carboniferous river deposition began early in the Riversdale geosynclinal valley, the outlet of the river system being through the Stellarton gap. These deposits are known as the Riversdale series. Riversdale deposition was likewise general in Cape Breton geosynclinal basins, but is seemingly absent in the greater part of the Cumberland valley. Lacustrine sediments are common in the Riversdale series, which is a very thick one, but no coals of workable size are known to occur, although climatic conditions had again reverted to pluvial. Thin carbonaceous seams, or only very thin coal, are sparingly present.

A disturbance of local importance, a third peak of accentuated uplift if measured by the positive areas, took place in the Riversdale epoch, but the event that chiefly concerns us was post-Riversdale warping, which brought about renewed deposition of river alluvium in the Cumberland valley. These deposits form the Cumberland series, which includes the Lismore formation or the 'Millstone grit' of the Joggins section and the Joggins or Springhill coal-bearing formation. The upper, or Joggins, formation oversteps the lower and overlaps upon the bordering uplands, a feature well displayed in the Springhill coalfield. Cumberland deposition was seemingly absent in the Sydney geosyncline, but took place in western

Cape Breton by reason of a river system that was tributary to the Cumberland valley. The Port Hood coal measures are Joggins in age.

Following the Cumberland series, there was a general disturbance and uplift of a more regional character, although new lines of folding were established, such as the Minudie anticline and the Malagash anticline. This disturbance was followed by an erosion interval which removed the Joggins formation over large areas of the Cumberland geosyncline.

Renewed accentuated subsidence of the geosynclines then brought about the deposition of the latest Pennsylvanian series, the Pictou series or the Morien series, as it has been called in the Sydney field. The Pictou series is present in all the geosynclinal basins. It was, moreover, predominantly a transgressive series, so that it eventually overlapped extensively upon the positive areas, even spreading in a widespread sheet over the old upland of New Brunswick. In the Cumberland geosynclinal valley, coal is sparingly present, although a thin seam is mined at Minto. In the subsidiary basin of western Cape Breton, the series is coal-bearing in the Mabou and Inverness fields. Study of the series in the latter basin should prove particularly illuminating, as it might indicate an overlap from the Sydney basin over part of the intervening positive crystalline area of Cape Breton. If such were proved or reasonably inferred, it would negative the supposition that the uppermost coal seams of the Sydney basin should be expected to deteriorate as a result of proximity to the Cape Dauphin crystalline mass. A study of the fossil flora of the Inverness coalfield should reveal the horizon of the coals there with reference to the Sydney section. Casual field observations of the flora made by myself certainly suggest a high horizon in the Morien series, and the stratigraphic relations on the west of the field are those of transgressive overlap.

A more direct economic bearing of the study of the tectonic movements and conditions of deposition such as outlined is instanced in the work already undertaken by Mr. Norman and myself this summer in the Port Hood coal basin. Those who are familiar with Mr. Fletcher's map may recall that a strip of rock outcropping on the eastern shore of Port Hood island is mapped as lower Carboniferous. A previous visit to

the locality had afforded me evidence that these rocks were Pennsylvanian in age, as conglomerate patches carried boulders with upper Windsor fossils. This season has confirmed this by the discovery of Pennsylvanian plants. These rocks, which are red in colour, are overlain with a local erosional unconformity by grey grits of the Pictou series. On the western side of the island, the Pictou grits overlap upon Windsor sediments. On the north shore of the island there is a fault between the lower red Pennsylvanian beds and the Windsor. Now the precise Pennsylvanian age of these red beds is of the greatest importance in connection with any further exploration of the Port Hood coals. It is, in fact, the crucial problem in this field, and fully as important as the flooding of the old mine. But the main point is this, that the evidence so far gathered is in favour of regarding the red beds as a local basal phase of the Pictou series itself and that, therefore, a new entry into the submarine area at greater depth is possible. Further prospecting for fossil plants, which are extremely rare in the red beds, may give the needed confirmation. But if not, it would be a simple matter to learn the truth by putting down a drill hole through the red beds close to the eastern shore of the island. For the most reasonable alternative to a basal Pictou age of the red beds is a basal Port Hood or Joggins age, and that would be ascertained by penetration into the Windsor beds at a probable depth of several hundred feet. If the age of these red beds is Port Hood there must be a major fault between the island and the mainland—a serious barrier to the submarine field. There is certainly no stratigraphic data in the form of dips and strikes that indicate such a fault, as the beds on the east side of the island have an altitude closely corresponding to those on the mainland opposite and the overlying Pictou grits are folded in a shallow syncline similarly to those on the mainland.

Another problem arises from the fault on the north shore, which may be of pre-Pictou age and which may cut off any accessible coal field west of the island; but this would not affect a submarine area in the channel between the mainland and the island over a frontage of nearly double the length of the two miles assigned it by Mr. Gray.

Before I close I would like to express my deep appreciation of Mr. Gray's contribution and my accord with his general forecast of what might be expected seawards in the Sydney Further field work would be necessary to test Mr. Gray's tentative hypothesis that the Cape Percy anticline may have tended to be a positive axis during the deposition of the Morien series or part of it, and may have influenced the sulphur content. The former hypothesis is fully in accord with the geological history of the other coal-measure basins, but whether the axis was efficient in modifying the sulphur content is another matter. Sulphur in coal that is directly inherent from the vegetable matter would not seem to appreciably exceed one per cent. Certainly there are not many coals with less sulphur, whilst the high-sulphur coals are very variable in their content of sulphur and usually carry a high percentage of pyrite, which may be finely disseminated. Pyrite-sulphur might be introduced subsequently to coal formation, in which case it would more probably occur along the cleat faces. Where disseminated, it was probably contemporaneously deposited by reaction between the decaying organic matter and sulphur and iron present in the swamp waters as soluble sulphates. The whole matter is worthy of attention, and this would require sulphur analyses by the chemists employed by the coal companies, which would differentiate between the varieties of sulphur present, the distribution of sulphur in a single vertical section of a seam, etc. It would require also the attention of the geologist studying the sedimentation, and it is a matter that had not hitherto drawn my attention. Already the suggestion comes to me that the possible outcropping of large gypsum masses near a coal swamp, for instance, in some of the positive axes, would bring much local sulphate waters into the swamps, and the result would almost certainly be deposition of pyrite. It is significant, for instance, that the Pictou coalfield, which is surrounded by an area in which Riversdale deposits might have effectually sealed the gypsum present, is particularly low in sulphur content, whereas the Joggins and Minto fields, close to positive sources of supply with thick gypsum accumulations, are very rich in sulphur. The sulphur present in waters draining a gypsum area would have an effect similar to sea-water encroaching on a coal swamp bordering a sea.

MR. J. P. COTTER: As a contribution to the discussion on Mr. Gray's paper, I have prepared some data in connection with in-bye compressors.

Two of the most serious problems in connection with in-bye installations are, first, the problem of a heaving floor, inasmuch as the movement is sufficient to destroy the alignment of the compressor and the effectiveness of the pipe line; and second, the problem of keeping the air cooled during compression, as the mine water, particularly in coal mines, is unfit for this purpose, and to pipe the water from the surface is rather impracticable. Some mention has been made of refrigerating plants in connection with the cooling, but this does not appear feasible.

From what information I could gather, it looks as though, up to the present date, in-bye compressors have limitations, and while some of my correspondents place this at 1,200 feet, I would rather be on the conservative side and recommend even smaller units, as they are more portable, more easily fitted, and have simple compression.

Another objection to in-bye compressors is that they limit the air storage, or volume of air compressed, to a much smaller amount than with compressors installed on the surface with larger air lines through the mines.

With the development of the long headways, requiring the installation of much larger drop hoists, the possibilities of these hoists depend entirely upon having a large piping system throughout the mine to act as a storage capacity, as the demand on the system is quite large for short periods, and the larger the volume stored the less fluctuation we have in the volume or pressure.

Assuming that 1,200 feet is the limit of the size for an underground compressor, it would mean one attendant for every 1,200 feet capacity, a rather expensive operation.

Any repairs that may be required will be somewhat more expensive underground than on the surface, owing to the time consumed in sending the parts and the men into the mine to make repairs. Furthermore, the upkeep is liable to be a little higher because the place is dark and the machine naturally would not be kept quite so clean.

The cooling of the water is another very troublesome problem. I do not believe there is anything in a refrigerating plant, as such a machine requires cooling water and would take a little more cooling water than the compressor would otherwise require. Commercial refrigerating machines are all ammonia machines, and I do not think that the mine inspectors would want to permit the extra hazard of the breaking of a connection carrying liquid ammonia, particularly in the mine. In brine refrigeration, the brine is not the refrigerating medium, which has to be gas, but it is the medium for transmitting the cooling from one place to another.

The refrigerating machine does not manufacture cooling. but transfers the heat to some place so that it can be thrown away. Down in the mine this heat can only be thrown away in cooling water, which is pumped to the surface, or in the mine air, which is drawn to the surface by the fan. Even if you could allow heat to be carried off in the mine air, this would mean a radiation system for cooling the compressor jackets on the compressors and for a compressor, say, of a medium volume at 3,000 cu. ft., I think that the system would be so extensive and expensive that it would more than eat up the saving in the air pipe-lines. It seems to me, therefore, that the practicability of the whole scheme depends on a supply of clean cooling water and pumping the discharge to the surface, that is, as far as the cooling is concerned. I consider it another serious objection that, to make a substantial saving in the piping, which is the only advantage in installing in-bye compressors, you will have to move the machine from time to time as successive sections of the mine are worked out. The moving and setting-up of, say, a 3,000 cu. ft. unit would be a very expensive item.

There are two objections to underground intake air. The first is that it may be very dirty, and would, therefore, have to be filtered. The other objection is that of the somewhat higher temperature of underground air. In the winter, the temperature of the underground air averages 60 deg. F., while the temperature on the surface averages 20 deg., which represents an 8 per cent increase in power for compressing the same amount of air underground.

The first objection that might be raised against surface compressors is the pipe-line resistance. This resistance decreases so rapidly with increase in the size of pipe that it is very easy to arrive at a size of pipe which will give a reasonably small loss of pressure. This loss of pressure is partly counterbalanced by increase in volume, so that the loss of power is only about one-half as great as the loss of pressure.

The second objection that can be raised is the leakage of pipe lines, and it should be quite possible to reduce this to a very small amount. Where compressed-air locomotives are used, the pipe lines are practically made bottle-tight for a pressure of 1,200 lb. to the square inch. I do not think that any ordinary movement of the floor would be liable to affect the pipe lines. If the movement is very considerable, there are new types of joints now used which are flexible, easily installed, and said to be absolutely tight.

The cost of 12-inch air line installed underground is approximately \$4.25 per foot. This extra cost of large pipe lines from the surface should be balanced against the following items:

Extra cost of compressors in small units instead of large units.

Extra cost of installing compressors underground.

Extra cost of air filtering system.

Extra cost of water cooling system.

Capitalized extra cost of attendance for small units underground.

Capitalized extra cost of power underground due to warm intake air.

MR. HERD: Mr. Cotter has given us his views regarding in-bye compressors in submarine mines, a considerable distance from the shore, and the gist of his remarks is that such installations are not satisfactory. There is considerable difficulty due to the movement of the floor. At present it is not so far underground to convey air and we must not forget that we have thirty or forty miles of air pipe on the surface.

MR. COTTER: The pressure has been tested to a drop hoist in No. 1-B, and in this instance the pressure dropped three to five pounds. Mr. Booth can tell us what the distance was.

Mr. Booth: Approximately three miles.

MR. James Purves: Mr. Gray speaks of the pressure due to overburden and says in effect, on page 5, that there is not much hope of relief. That may be true of the south side of Sydney harbour, but I do not think it is altogether true of the north.

In Princess there is every indication of flattening out in the direction the deeps are pointing. The first 10,000 feet dips at an average of 4°50', the last 3,400 feet averages about 3°. With the sea-floor dipping 2°, there is only now 1° of gain in the cover. I think that the extension of these deeps another 3,000 feet may result in the dip flattening to the point where no gain in cover will be experienced. Possibly at a mile it may be found that the cover may be reducing. This condition to a lesser extent is noticeable in Florence colliery.

Mr. Gray's statement would lead one to think that nothing can be looked for but an increasing overburden. Dr. Bell is, I think, of the opinion that the direction of the haulage slopes in the collieries on the north side of Sydney harbour may be heading across the basin formed by one of the minor folds indicated in this instance as the basin lying between Lingan anticline and the Boularderie anticline, and that at a point not too far distant these slopes may be beginning to turn up or continued in the same direction.

The weight of the overburden for 1,400 feet cover, with roof and pavement condition as in Princess, results in a serious addition to maintenance costs, and while this has not reached an unbearable figure yet, the net result will mean a continued fixed charge on air and haulage ways.

I think we can go some distance seaward yet, before the point of economical failure is reached, due to increasing overburden; and I think we are entitled, in the instance of Princess colliery at least, to hope for relief rather than otherwise from a possible or probable flattening out of the seam.

I would suggest that Mr. Gray consider this and possibly modify the statement referred to in his excellent paper.

MR. JOSEPH KALBHEEN: Mr. Gray has stated that the successful mining of the Cape Breton coal seams in the future depends on a number of factors, among them being depth of

cover and consequent roof pressure. Support and maintenance of main roads and airways, is, therefore, a problem to be solved by the mining engineer, and a brief article on this subject is in order in discussing Mr. Gray's excellent paper.

I wish to discuss some of the methods of roof support adopted in mines having heavy overburden, and this from the technical viewpoint only, ignoring for the time being the economic aspect of the question, although I may be permitted to say here that frequently, or rather generally, heavy initial expenditures in the way of roof support are justified by the results obtained. As a proof of this statement I will submit figures later on.

It must be admitted that the present system of timbering and supporting the main arteries of our pits will, with increasing depth and heavy overburden, have to be abandoned. Some of them have already reached the stage where maintenance with wood or steel rails is inadequate. I am referring here more particularly to the New Waterford collieries. More substantial methods have to be adopted, and in search for such we turn naturally to the European countries, where conditions similar to, and more severe than, ours have prevailed for many years.

The driving of air and road ways in rock measures other than coal is not unusual where the adjoining strata to the coal seam are weak. Where tunnels are developed in rock strata, the walls and roof are frequently lined with 'gunite', corrugated iron, brick, concrete, or reinforced concrete, in many shapes and forms. The adoption of one or more of these systems depends entirely on the forces to be counteracted.

If the rock in which the tunnel is driven is in itself able to withstand the pressures, but is liable to deterioration from the action of the air, then an application of gunite can be considered adequate; but a lining of concrete of moderate thickness would be preferable, in the writer's opinion, especially as the use of scrap rails or condemned ropes in the concrete would increase the strength of the concrete beyond belief.

In a stratum where moderate, uniform pressure is manifest, a lining with brick has proved successful, but in recent times concrete is coming more into favour.

Dimensions and the filling of all voids are the important points to be observed.

In all cases where the strata are subject to irregular pressure, bending stresses appear and, to overcome this, reinforced concrete has been used with success. The superiority of reinforced concrete is attributed to the fact that all tensional stresses are taken up by the iron and all compression by the concrete.

The ability of the reinforced concrete lining to withstand the pressure depends on the sectional form of the lining. This may be either the open or the closed type. The different open systems provide protection for the roof and the ribs only, no protection being made against the upheaving of the pavement. Such a system is evidently wanting in case of bending stresses. Experiments, and in some cases very costly ones, have been made to overcome this want, by increasing the thickness of the concrete, but it was found that the increased thickness was detrimental and had a reverse effect on account of the rigidity of the supporting walls.

It is therefore of vital importance to pay due consideration to all stresses when selecting the system, and a section which will adapt itself best to resist all anticipated forces should be chosen. Either a semi-elliptical or cylindrical construction will best accomplish this.

We have then the closed form, which protects roof, ribs, and pavement. The protection of the pavement prevents any dislocation of the rib walls and the heaving of the bottom. To provide the necessary elasticity to the supporting member, three elastic joints—two on the bottom and one on the apex—should be inserted. Such joints may be constructed either as cylindrical joints in the concrete or as squeeze joints made from wood. Under very unfavourable conditions the joints may be considerably increased. This serves the additional advantage of facilitating eventual necessary repairs. Such lining may consist of either well designed previously cast segments or be put up *in loco*.

The former method has also the advantage of easier removal in case of repair or for use in other parts of the mine.

Greater difficulties were experienced where different trunk lines join, *i.e.*, on pit bottoms, landings, etc. Here the pressure effects from both sides have to be added and a much stronger support must be provided. A combined building of iron concrete and wood has helped to overcome this problem. As previously stated, the support must not be too rigid—enough elasticity must be provided to overcome sudden, heavy forces. To obtain this, a cushion, consisting of a layer of sand, ashes, or wood, inserted between the lining and the strata, and elastic joints in horizontal and vertical directions, will ease off any sudden impact.

Another system quite frequently adopted in exceptionally heavy ground is the Breil compound system. Here two cylindrical steel girders of different diameters, made up of several segments tightly connected with radially located angle-irons and horizontally cross-wise arranged flat-irons are imbedded in concrete. Size of iron and steel and thickness of concrete depend on the desired strength.

A lining of this kind was adopted in the Mathias Stinnes mines in 1918. The landings in the pit bottom got under creep. Arched brickwork with elastic wood joints of 3-foot thickness gave out.

One hundred repairmen were needed daily to barely keep these roadways enough open to permit passing of boxes. The management, as a last resort, decided to try the Breil system and secured 420 feet of roadway at a cost of \$66,000. The cost for previous maintenance amounted to \$385 daily. It took also only 171 working days for the amortization of the initial expenditure for the new protection. The Breil system arrested the movement of the strata, and up to date practically no expenditure has been necessary for maintenance.

MR. T. L. McCall: I have been very interested in Mr. Gray's paper and also in the discussion on it. Of the latter, my attention was centered principally on the remarks of Mr. Gordon, for ventilation is one of the first problems that has to be tackled when mining under sea at long distances from the outlets. In thinking the matter over, I reached the same conclusion that he did, namely, that it might be necessary to pump air into the workings at high pressure and there

release it for ventilation purposes. However, in this age of rapid advance of science, it is quite possible that some catalyst may be found which would render the liquefaction of air a cheaper proposition than it is today, and if such were the case it might be then feasible to convey the air into the workings as a fluid and once there expand it and use it for ventilation.

MR. HERD: The possibility of using liquid air may not be so far in the future. In fact it has already been attempted in one section of a mine in Germany but was abandoned due to the cost. However, it will stand investigation, and as science improves it is just probable that it may be economically applied to under-sea mining.

MR. McEachern: Severe criticism has been levelled at the work of the mining men of the past, but the fact is lost sight of that these men had just the same or greater difficulties than we now have, in getting money to do the work. This shortage of capital often compelled those men to take coal where it was the cheapest to mine and the easiest to get. This, in many cases, no doubt, now makes mining operations on an extensive scale very difficult. Take one case, for example, which I have in mind, where pillar work had to be discontinued because of superincumbent pressure. As an experiment, places were first driven 60 feet wide and afterwards reduced to 22 feet, and packed in the centre. After the gases had escaped and the roof settled down the roadways were brushed. same may be applicable to the conditions of today because of heavy roof which has been affected by gas given off under great pressure, as well as by strata settlement.

MR. HERD: After the settlement and exhaustion of gas, and the place was brushed, you would then put in your steel and concrete supports?

MR. McEachern: I certainly would, where necessary.

MR. HERD: We have to do some experimenting before we can decide on the best methods of working.

MR. H. B. GILLIS: I was very much interested in Mr. Cotter's discussion relating to in-bye compressors. Power loss on small air lines prompted us to carry on an experiment at Wabana a few years ago. The conditions prevailing were: high coal costs, single shift operations, and long air lines. At

the time a number of small underground hoists and some of the pumps, in addition to all drills, were operated by air. The heaviest demand on the compressors occurred over a period of three hours, and during that period the pressure at times dropped from 70 lb. to 40 lb. Throughout the remainder of the shift, or for a period of seven hours, there was ample air, at good pressure, for all purposes.

The object of the experiment was to equalize air pressure throughout the entire shift. To accomplish this a section of the mine workings, at the bottom, was sealed off and used as a reservoir to store the air during the time the power plant was operating. This air was forced into the chamber and displaced water, which was forced to a sump at a higher elevation, which maintained a fairly constant level and a constant pressure on the air in the air reservoir. The experiment, as far as it was carried, indicated that this plan could be used successfully. Tests made during the experiment showed about a five per cent loss in storage, but it was believed this could be materially reduced. If a storage of this nature could be arranged in the coal mines, near the working face. it would to a large extent offset losses due to long pipe lines. For the same size of pipe a greater volume of air could be delivered at high pressure to the face, as peak loads would be handled from the reservoir and thus decrease friction loss in the lines.

In regard to Mr. Gray's paper, I must compliment the writer on the success it has been. He has outlined our development of submarine mining and touched on the vital points that must be settled before a comprehensive plan of recovery can be adopted for the submarine areas. His description of the structure of the coal basin and the origin of the sediments is, I presume, not to be accepted as a final judgment, but rather as indicating the importance of geological research before our operations are much further advanced. It is highly important that the coal horizon should be determined and any structural difficulties of the basins worked out before final plans for the recovery of the coal are undertaken. We cannot hope for exact knowledge, such as could be worked out for land areas, but a study of the shore line from cape North to point Aconi, together with test drill-holes on Flint island and

Scaterie, should furnish information that would no doubt be valuable to those who must plan future operations in the The magnitude and cost of a modern colliery make it imperative that all information possible should be assembled so that the best site for maximum recovery should be selected when future submarine workings are projected.

Mr. RICHARD KIRKBY (1) (East Wemyss) (2): Mr. Gray has given us a paper which is of the greatest interest to mining engineers and geologists. It contains an enormous amount of information and will be accepted as a reference book by all

who are interested in undersea coal mining.

Undersea mining is far more fascinating in its problems than land mining. I know something of the Sydney coalfield, as I was engaged there for a few years and I am acquainted with the Firth of Forth field from both shores. The same problems exist there as in Sydney, and as they must do in all undersea coal deposits. Three of these are:

(1) How far do the workable coals extend?

(2) What are their quality and thickness?

(3) What depth do they attain?

Such questions may not be fully answered here in Scotland or in Cape Breton even a hundred years hence, and when studying such a subject one almost feels one would like to emulate Methuselah. if for no other reason than to know what the conditions are a few miles under the ocean.

Mr. Gray has, I think, discussed every point, and certainly all the important ones, in connection with the layout of future workings in undersea fields, and if I go over some of the same questions it will probably be invariably in the way of corro-

borating his opinions, as I think these are very sound.

Referring first to the question of water, it goes without saying that every possible precaution should be taken to guard against sudden inbreaks of water. As Mr. Gray says, only one serious accident has happened by the sea breaking into colliery workings, namely, at Workington in 1827. Another accident somewhat analogous to the Workington case occurred at Landshipping, in Pembrokeshire, in 1843, when the neighbouring river broke in and drowned forty men and boys. It

⁽¹⁾ Agent and General Manager, Wemyss Coal Company, Fife. (2) Contributed discussion, read by Mr. W. Herd.

is evident, therefore, that with these two 'exceptions to prove the rule', undersea mining has been conducted in the past with the greatest care for the safety of the men and the mines. With careful underground surveys, horizontal and vertical, and taking due account of the surroundings in the sea-charts, there is very little danger of getting a sudden inrush of water when ordinary precautions are taken. There are one or two conditions in the sea-bed which one should be apprehensive of in certain localities. For instance, one cannot always rest assured on some coasts that the sea bottom shown on the chart is a solid bottom. If there are any land indications which would lead the mining engineer to think there had been a deep river bed in the glacial age, the sea-bed ex adverso, that place should be explored. When I was in Cape Breton some years ago I was just a little doubtful of the actual position of the hard bottom in Indian bay between Dominion No. 1 colliery and Lingan. It might have been the case that sand extended for a few hundreds of feet depth under the sea-bed.

A special raft was constructed and this was anchored at five different points from a quarter to three-quarters of a mile from the shore, the exact places being given by a theodolite set up on shore. Bores were put down through the soft clay and sand until hard rock was met. The results were very reassuring, as the rock head was found within three to five feet, I think, of the sea bottom. The work could only be done in calm weather.

Again, if there are signs of basaltic dykes inland, these should be suspected in the sea bottom, and exploring places should be driven ahead of the workings. Where such a dyke is struck at a fair depth it is seldom much water is got, but I once struck one where the seam was 600 feet deep and, as soon as the face of the basaltic wall was bared, a feeder of pure sea water came in at the rate of 150 gallons per minute. We suspended operations until a larger pump was installed and then went ahead to drive a mine through the dyke. The feeder did not increase, and when the dyke, which was 40 yards wide, was pierced, strange to say there was practically no water got from the inner face. The 150-gallon feeder gradually pined off. Evidently the water in its course carried sand with it, which ultimately practically filled up the crevices

between the sedimentary rocks and the vertical dyke. This dyke was met with on the south side of the firth of Forth. Sea water may be allowed to get into the mine workings, however, in quantities which, while not dangerous to the safety of the mine, may, nevertheless, be sufficiently large to add materially to the cost of pumping during the whole life of the colliery. Unless it can be definitely ascertained that the undersea outcrops of coals and their overlying sandstones are well covered with a good thickness of boulder-clay it will pay well to leave very broad strips of unworked coal next to the outcrops until the colliery is nearly exhausted. It is not only the quantity of water which matters, but also, of course, the depth from which it has to be pumped. It takes approximately one unit of electricity or its equivalent to pump one thousand gallons of water against a head of one hundred feet, and of course it will therefore take ten units to pump this amount against a head of one thousand feet. In steeply inclined workings it is not always easy to prevent water from travelling still further to the dip, but every attempt should be made to do this.

Coming now to the methods proposed for future undersea winnings, I agree with the author that deep shafts and cross-measure drifts will make for economy in pumping water and hauling coal. It is, of course, sometimes the case that cross-measure drifts tap more water than is got in the ordinary working of the coal, but nowadays this water may be sealed off by cementation where the quantity justifies the expenditure. I think these mines should rise in-bye at the rate of about one in two hundred to facilitate free drainage. My own opinion, too, is that two tunnels, each with a sectional area of one hundred square feet, are preferable to one with, say, two hundred feet area. I think they would be cheaper to make, and the advantage of two mines in place of one is obvious. There would therefore be four tunnels, two intakes and two returns, for a large area of work.

The subject of the correct order of working superimposed seams will be debated, I suppose, to the end of time, because it all depends on circumstances and conditions which is the best line to pursue. However, if it were possible to say that, for all practical purposes, all the seams in a particular area were

alike in quality, then I think it will be granted there is one proper order of precedence and one only, namely, to work the lowest seam first and follow on in rotation. The advantage in this is that the next seam above may be attacked just as soon as the subsidence due to the lowest seam is settled, and two or three or even more seams might be worked simultaneously at certain distances one behind the other.

If a higher seam be worked first, then a lower coal must either wait for the exhaustion of the upper one or the subsidence from it will damage the main roads and water courses in the upper. I cannot just say I know of any series of coal seams which are all of the same quality, but at all events the proposition is a basis to start from in the discussion of the problem.

In conclusion, I would like to emphasize what the author says regarding the wonderfully good natural conditions of the Sydney coalfield. For quality and thickness of its seams, for regularity and the freedom from faults, and for easy gradients, there must be few fields of equal extent to compare with it, although I must admit I know very little regarding the coalfields in the United States.

What has always struck me as strange is, that there has never been a case of spontaneous combustion underground in this coalfield, although two of the seams, the Phalen and the Hub, are thick coals. Altogether the field is almost ideal. It would only be better (financially) if it were laid out on the surface suitable for quarrying, but then its interest to the mining engineer would be totally gone.

MR. HERD: We are much indebted to Mr. Kirkby for a contribution out of his fund of valuable experience. After all, submarine mining is not so very different from land mining except that greater care against subsidence must be guarded against in submarine mining at light covers. In submarine work, transportation and ventilation will be the big problems. I am sorry that Dr. Bell is not here to say something about the probable deposition of the Sydney coalfield. Mr. Gray suggests the coalfield was deposited by a flow from cape Dauphin and that the lower seams are not as persistent as the upper ones in the direction furthermost from the point of supposed origin. At Springhill the seams appear to have a similar type of deposition.

MR. F. W. GRAY: I am extremely gratified that such nice remarks have been pased about my paper and at the intense interest shown by the discussions. Much thanks are due to Mr. A. L. Hay and the engineering department for the valuable assistance given me. If this paper did no more than to cause us to 'furiously' think, it has achieved something. The preparation of it was an education to me and it took ten months to complete it.

Mr. Nicholson mentioned that the transportation of men is an outstanding question in remote undersea mining. From what I can find, we are really ahead of other fields. They do not approach us in that regard in England, particularly in the Durham fields. We are well in the lead and our system is

safe, and we have to consider that as well as speed.

In speaking of Port Hood and the number of pillars drawn there, I said 'one', but Mr. McEachern declares there were five. I will see that the necessary correction is made in my paper before it appears in the Transactions. Whether the pillars were drawn or robbed, or there was a crush or a creep in the pillars above No. 5, seems to be a debatable question. Mr. McKenzie was kind enough to say that a lot of information about the Port Hood case is available, and I would suggest, if it can be put in shape, that it be published in the next Mines' Report. There seems to be some misconception of my remarks about the Port Hood occurrence. The report of the Commission conveys the sentiment of the Commissioners. report was made nearly twenty years ago and there are some features that possibly would have been referred to, had the Commissioners not considered the possibility of injuring the financial standing of the property. I like to see discussion, as this is no mutual admiration society, and a discussion sharpens the wits.

Mr. Rorison has supplied me with a few particulars

regarding this matter, as follows:

"When surveying the mine that was worked by the late Malcolm Beaton, immediately to the south of the old Port Hood mine, in the year 1920, the level (the only one that was driven in his mine) was driven through into the main slope of the old Port Hood mine. He pumped the water clear of his mine with a 100-gallon capacity

pump and kept it down 50 or 60 feet below his level, which was 300 feet down from the surface. Mr. Beaton and myself, that day, took measurements during high and low tides and, the pump standing idle during the time, and we found no difference in level of the mine water, nor any tide effect.

"Same result was found this month on the 11th of June. No tide effect in the mine, and water in the slope has receded 30 feet on the pitch. Water being pumped out of a small mine immediately to the north of the old mine; same pump is being used now that was used by Beaton".

The dispute seems to be about the point of entrance of the water. My contention is that it did not come from the vertical crack, but it seems I cannot convince Mr. McEachern and Mr. McKenzie of that. My opinion is that the water entered between high and low tide, as evidenced by the ebb and flow in the mine. The protection of undersea outcrops referred to by Mr. Kirkby is being pretty well looked after. An investigation of the water coming into No. 1-A mine in the Sydney field showed that it originated from a feeder in Lingan bay, and I think this is what induced the borings Mr. Kirkby mentioned. Every other inundation, those in Chile and at Mabou and Sydney Mines, came about through the driving of levels rising towards the crop. The tendency of the water is to choke itself, as stated by Mr. Kirkby in referring to his experience with a basalt dyke.

The discussions by Messrs. McMahon, McEachern, Gordon, and Kalbheen are supplementary to, but somewhat beyond the range of, my paper. I have been accused of having a mind too historical and dwelling too much on the past, but this can not be said of this discussion. We are obliged to Mr. Cotter for his contribution to the discussion and to the Ingersoll-Rand people for taking the cue with reference to underground air-compressors. The people who have machinery to sell should be in on the ground floor, and it is a pleasure to us to see present at these sessions, representatives of the people with whom we do business.

It has been suggested that air-filters would be a necessity underground. I think it will be found that there are many places underground where there is less dust than around some of the surface plants.

Mr. Purves gave instances of changes of gradient. I do not think there is any doubt about the measures flattening seaward and attention is drawn to this in the paper (1). Dr. Bell mentions the geosynclines as being basin-shaped and coming up on the farther side. Before it takes the upward curve it will likely run flat a long time. I do not think the overburden will increase indefinitely.

Mr. Gillis has mentioned conditions at Wabana. I hope that he will write a paper complementary to mine, as he has made a study of the geology of Wabana.

I tried to show on the diagrammatic plan the structure of the Sydney field, and the main point of this study indicates that the maximum point of depression is in a line running from Table head to Schooner pond. If No. 6 was put down to get the full 'take', it could not be located in a better position. The future, I think, will prove that the main part of the coalfield is off Schooner pond and Flint island. I have been accused of not giving due credit to the men of old, but I hope that such a conclusion will be dispelled. In an article recently written for *The Halifax Chronicle* I gave full credit to these pioneers, but it must be remembered that my paper was written for a wider circle. In conclusion, I desire to thank Messrs. Herd, Hay, and Mifflin, and all those who assisted.

Colonel McKenzie: I have read Mr. Gray's paper, which I think is most comprehensive, and I have listened to the discussion with much interest. I have no doubt these discussions will help us out in our work in the future. Colonel Cantley told you of the first underseas mine in Nova Scotia, which he mentioned was started in Stellarton under tidal water by the General Mining Association in 1826, and a large percentage of the coal extracted from the Foord seam since that time has been transported underground beneath the East river. At the present time 90 per cent of the output from the Allan mine is taken to the shaft under tidal water.

⁽¹⁾ See page 21.

MR. GRAY, in conclusion, read an extract from Brown's Coal Fields of Cape Breton, to the effect that the General Mining Association was organized in 1825 and that Mr. Brown was sent out in the spring of 1826 to make a survey and report of the coal fields of Nova Scotia and Cape Breton, and the Sydney Mines' areas came into the possession of the Association on the 1st of January, 1827. Commenting on this Mr. Gray said that apparently Col. Cantley and himself were both right in their references to the date the General Mining Association started operations.

CONTRIBUTED DISCUSSION

MR. Walter Herd: One cannot help being impressed by the opening statement in Mr. Gray's paper, that 25 per cent of the total output of Canadian coal comes from under the sea. This seems to loom somewhat larger in the mind than the further statement that 55 per cent of the annual coal output in Nova Scotia comes from submarine areas—I suppose, to some extent, because of the disparity in the published coal reserves of Nova Scotia and the great western coalfields, more especially of Alberta. To Nova Scotia itself the economical winning of undersea coal is of paramount importance, but it assumes an importance beyond the confines of the Province when it is realized that Nova Scotia produces almost 30 per cent of the world's undersea coal.

The special conditions governing the extraction of undersea coal have been ably reviewed by Mr. Gray, and I will briefly recapitulate them for the purpose of discussion. These are:

- ese are:
 - (1) provision against inundation;
 - (2) permanence of main road and airway construction;
 - (3) ventilation;
- (4) transportation, not only of the mineral produced but of the workmen;
 - (5) transmission of power;
 - (6) roof control; and
 - (7) assured tenure to large leaseholds over a long period.

(1) Provision against Inundation:

The importance of this has been stressed and we have only to consider the disasters of the past, inbreaks of the sea having occurred on the east and west coasts of England, in Japan, Spain, Chile and Australia, together with two comparatively recent instances in Nova Scotia—at Mabou and Port Hood—in all but one case the result of mining too close to the sea bottom.

Just what would be the safe minimum distance to establish will be governed by local conditions, principally the composition of the strata overlying the seam or seams to be mined, the type of sea floor, and the presence of faults. Legislation restricting the distance a seam may be mined below the sea varies in different countries, Chile and Australia stipulating 120 feet, Britain 270 feet, and Japan 300 feet, while in Nova Scotia it is 180 feet. Only in the case of Britain, where generally speaking the circumstances governing each case are considered, are there definite regulations governing at what depth the entire seam may be extracted.

It has often occurred to me that the Nova Scotia law and that of several other countries is deficient in stipulating a definite cover without reference to the possible mining of underlying seams. While 180 feet may be ample in the case of a single seam, this may not be sufficient in certain cases should another seam be mined underneath, and I have recommended a minimum cover of 200 feet in the specific instance of mining the Harbour seam, under which we knew that within a few years the Phalen seam would be mined.

In the Sydney district, where the sea floor is regular and is not overlain by any depth of sand or alluvial deposits, where strata are practically free from faults and the overburden contains fully 50 per cent of shales, the remainder being beds of sandstone and a few thin clay bands, 180 feet appears a reasonable minimum cover under which to extract 50 per cent of the seam, provided adequate pillars are left in an underlying seam along this cover line. The need for accurate soundings and levellings is evident.

The depth at which it is prudent to totally extract a seam is more difficult to determine and may be governed by many factors. Assuming strata conditions such as obtain

in Nova Scotia, the chief of these factors would be, what overlying seam or seams have been worked, and how? We would unhesitatingly totally extract a single seam, say five feet thick, at 800 feet or over solid cover, provided some sort of packing to ease down the roof was adopted. At what depth a lower seam should be totally extracted under the waste of an upper is open to much argument, but I would advance the view that if an upper seam had been totally extracted, by which I mean no stumps or pillars left, then the depth at which a lower seam might be totally extracted would be arrived at by assuming the strata above it intact but its thickness increased by one half the thickness of the upper seam extracted.

It is almost futile to discuss the order in which superimposed undersea seams should be mined, because, in the future as in the past, it is evident the law of economics will prevail and each generation will take the most profitable. In the case of two or more seams of nearly equal value, where the output desired could be obtained from one of them, then it would appear advisable to mine the upper first, main roadways only being developed in the lower as the upper approached exhaustion. The roadways in the under seam would be undisturbed and the upper seam would be subject to no subsidence with development roads only in the lower. I would also work the upper seam first in the case of two or more very thick seams in close proximity, as working the lower first might seriously damage the upper. In cases where an output was desired from the working of two or more seams simultaneously, the lower should be kept in advance, a distance which would allow of a lapse of time necessary to consolidate the overlying strata; otherwise subsidence would cause serious havoc to the roadways in the seam above. The time it would take to consolidate the strata would depend upon the thickness of the lower seam, the distance between seams, and the mode of work. Three years would probably be an average lapse of time necessary in the conditions met with in the Sydney coalfield, assuming longwall workings well packed. Tight packing of the waste, apart from preventing sudden subsidence, furnishes an added safeguard in that it induces a longer draw, making a forward break with a small

angle, which break the overhead weight would tend to immediately close and prevent an inflow of water, as against the almost vertical break from the unpacked waste of a thick seam.

In discussing the mining of superimposed seams, on page 109, Mr. Gray records information furnished him by Mr. Jas. Cumberford, relating to the Schwäger collieries in Chile, where the experience was "that, with the upper seam in advance, the lower seam coming behind disturbed the roadways in the seam above and the lower seam was harder to mine". That is what one would expect, but in the following sentence, where it is stated "that with the lower seam worked in advance there was slight effect only upon the mining of the upper seam, but the roadways in the lower seam were disturbed", I do not appreciate why the lower-seam roadways should be disturbed, considering that the seams mined were thin and surrounded by soft strata. I offer the suggestion that the movement noted was due to the soft floor of the underlying seam and would have occurred had the upper seam remained intact.

I believe, in the minds of many, the effect on an upper seam of mining a lower is exaggerated. My own experience is that in most cases it is negligible, when an interval of anything over 200 feet exists and, of course, assuming seams not thicker than those found in the Sydney coalfield.

A study of subsidence on land caused by extraction under different methods and at varying depths is of first-rate importance as an indication of what may be expected undersea.

The greatest immunity from inundation would be afforded by some form of hydraulic stowing, but the cost of this added to the already onerous conditions of undersea mining is at the present time prohibitive.

Barriers.—The policy of leaving adequate barriers between adjacent submarine mines is sound, and these barriers must be considered vertically as well as laterally. In the case of undersea mines which are an extension of land mines, it will generally be found that the barriers left under land are not large enough (in some cases they do not exist) to withstand the pressure from inundation of the sea on either side, resulting in the flooding of one mine extending to those

adjoining. At considerable depth the size of an effective barrier might be very large, depending to some extent on the nature of the coal and adjoining strata. This would result in a very considerable loss of coal, and although it might be argued these barriers could be recovered in retreat, generally the roadways giving access to them would be non-existent and in practice they would probably be left. I would suggest, if at all feasible, that barriers be formed of hydraulically-packed sand. They can be made of ample size without loss of mineral.

In the case of vertical barriers, it should be arranged that superimposed seams in adjacent mines do not overlap unless the distance between the seams is such that the flooding of one would not affect the other.

In undersea mines it is desirable to have ample pumping capacity, because it not infrequently happens that at faults, or in driving crosscut-drifts, considerable quantities of water are given off temporarily and unnecessary alarm is caused by the drowning of inadequate pumps.

(2) Permanence of Main Road and Airway Construction:

Within recent years the practice is an ever increasing tendency towards permanence in the construction of main haulage-roads and airways in the case of the larger and deeper well-equipped land mines. Experience has proved that the more or less temporary support given to main roads in the past cannot be economically applied where a certain depth is exceeded; that in the ultimate it is far cheaper to face a comparatively heavy first cost in more permanent construction than to keep a constant repair force employed, apart from a more secure continuity of output. If this is true for land mines, it is much more essential under submarine conditions, where it is impossible to sink shafts ahead and forget the lack of vision in the past. Although submarine mining must bear costs of the future, these are often more apparent than real. It may generally be said that the distance it is possible to mine seawards is to a large extent governed by the permanence of construction on main roads and airways and the vision to design these of adequate size. I agree with the author that economic and not physical conditions will be the limiting factor, and that the coal seams in the Sydney field extend beyond our ability to mine them under presently known methods. One is, however, struck by the number of instances cited where submarine mining has ceased due to deterioration of the seams seawards.

(3) Ventilation:

Of all the problems connected with the design and layout of a large submarine colliery working a seam giving off gas, that of providing adequate ventilation for eventual needs is probably the most important. It is to a large extent inseparably connected to an efficient transportation system.

It will generally be found that, due, first to the economic necessity which causes only the workings of the thicker seams to become extended seawards and, secondly, to the need for supporting the sea bottom till a sufficient cover had been reached, the system of mining for a considerable distance seawards in most cases is by the room-and-pillar method. In nearly all instances the main airways developed as normal places, and no heed was given to the ultimate large number of stoppings which would be necessary. The practical impossibility of keeping these stoppings tight under so many varying conditions and ground movement from one cause or another results in a loss of air often so serious as to limit output at no great distance seawards. The author gives the specific instance of the difficulties encountered in Whitehaven from this cause.

While, of course, it is possible to drive additional airways in the strata above or below the seam, this is not always economically feasible, and often the best has to be made of existing conditions; but too much stress cannot be laid on conserving the air as far into the workings as possible, which can only be done by keeping stoppings tight. They should always be visible for inspection. It is also imperative to keep the airways of as large dimensions as feasible, and if the economics of the situation will not allow of driving and maintaining additional airways eventually necessary, a reservation should be left for this purpose.

It is possible to do too much for posterity, and making provision for certain eventualities too far ahead with our less enlightened knowledge may, in the future, be found to have been unnecessary. Ventilation in the future may be derived from some allotropic form of element yet to be discovered or still in the experimental stage.

Enough would, however, appear to be known on which to base a ventilation policy for submarine mines under the conditions found in the Sydney coalfield. As previously stated, ventilation and transportation are to some degree interconnected, and looking at these two problems when considering a future layout it would seem that the best results for present and future economic winning of undersea coal will be obtained by sinking shafts below the coal seams and driving nearly level cross-measures drifts seawards to intersect the seams. Under this system, the intake air would be confined to the solid-rock tunnels and delivered without loss to their intersection with the coal seam, the return being in the seam. The only drawback to this method is the large initial outlay involved, but it appears to be one which, over a period of years, would give adequate returns.

As an alternative, and one which would eliminate leaky stoppings but not help the transportation at great distances, is the suggestion I have heard put forward that strips on either side of the airways be hydraulically stowed with sand, or that the airways themselves be formed in a sand-stowed area.

(4) Transportation:

With seams dipping seawards at even the moderate grade of 10 per cent, the problem of transportation of large outputs becomes serious when a distance of three miles is exceeded. Up to that point endless haulage can deal with 2,000 tons per shift, but, to continue this output for greater distances, costly transfers become necessary, the possibility of breakdowns is doubled, and the transportation of men must be by separate roads, involving further transfers and added datal force.

Mr. Gray has mentioned the suggestion of the late Geo. Blake Walker, made some twenty years ago, that the submarine coal in the Sydney coalfield should be won by a deep shaft and cross-measure tunnels. I believe the lapse of

time has fully confirmed this view, and today it is more apparent than ever, not only from the transportation viewpoint, but also, as I have pointed out, to establish effective ventilation.

The rapid transportation of men and minerals by means of electric locomotives running in almost level tunnels means that an extra few miles adds very little to the cost. With well-laid track and properly designed mine-car bearings, 25 to 30 miles an hour need not be looked on as an excessive speed.

The author gives a sketch to illustrate the extreme application of the deep-shaft and the cross-measure-tunnel development idea, suggesting a deepening of the No. 2 shaft of the Dominion Coal Company at Glace Bay to 2,500 feet and drifting level seawards. Such a drift would cut the Phalen seam at about 4½ miles from the shore, and the Hub at 8 miles. While this would tap the various seams at a depth which could not be considered excessive, and electric-locomotive haulage would solve any transportation difficulty, the great first cost, apart from a ventilation problem also involving cost, must debar an ambitious scheme for this generation, under existing competitive conditions.

Something which does appear to me as not only feasible but highly desirable during the present decade is to sink a shaft about 1,800 feet at the location suggested by the author and set away drifts seawards rising 1 in 200. Such drifts would cut the Phalen seam 31/3 miles from shore, the Harbour $4\frac{1}{2}$ miles, and the Hub $5\frac{3}{4}$ miles. For some considerable lapse of time the drifts would stop at the Phalen seam. As the area tributary to that seam became exhausted, these drifts would be extended to the upper seams, and, in order to limit dip haulage in any seam to a minimum, an auxiliary shaft could be sunk about the point of intersection of drift with Phalen seam, say to a depth of 800 feet below that seam, and drifts started from that point seawards. tap this seam fully 2 miles further seawards, or 5½ miles from shore. The introduction of a transfer would add somewhat to the cost, but the interest on the large initial cost to reach the seam at this latter point by an originally deeper shaft would be saved for many years and would in the ultimate more than compensate for the transfer cost. Such a system could be repeated, the benefits being level haulage for mineral and men, and ventilating current delivered without leakage to point of intersection with seam.

(5) Transmission of Power:

The transmission of power need offer no serious difficulties. Where advisable, electric current would be used, high-voltage cables being laid in the companion drift.

Should it not be deemed prudent to use electric power in the inbye workings, I believe compressed air should be generated on the surface, and not underground by electric compressors as suggested. With large air-mains, the loss in transmission is small and under certain conditions can compare favourably with electric transmission. This has been demonstrated in France and at the Powell-Duffryn collieries in South Wales. Although the cost of several miles of air-pipe is high, against it must be placed the cost of the extra cable-capacity for underground electric compression, and I fear that ground movement, especially where more than one seam was worked, would cause endless trouble, which would be added to by cooling and air-washing difficulties with large compressorunits underground.

(6) Roof Control:

The control of the roof under submarine areas will be the same as on land, with similar conditions of overburden, with the possible difference that, due to the prior mining of certain seams under the sea, it may not be advisable for possible inundation reasons to totally extract subsequently-mined seams in close proximity to these 'wastes', and some form of room-and-pillar or modified longwall, leaving in supporting pillars, may be necessary, increasing at depth the difficulty of roof support or aggravating the tendency of a soft floor to heave.

(7) Assured Tenure to Large Leaseholds over a Long Period:

The usual system in Nova Scotia of issuing leases of a not greater extent than one square mile, and for a period of only 20 years, is eminently unsuited from the economic

standpoint for the best layouts and the ultimate maximum recovery of submarine coal under the conditions existing in the Sydney coalfield. True, the 20-year leases are renewable, but there is no guarantee of such on the same terms, and it appears to me that on certain agreed terms 20 years is altogether too short a period to justify the very large expenditures necessary to equip a mine with the object of ultimately winning remote undersea-coal. This period might easily elapse before it was possible to reach outer leased-areas, which, due to the limitation of one square mile, must for protective reasons be acquired by the concern financing the operation for the The reasonable extraction of the not too remote areas. solution for undersea leases would seem to be in the case of minerals owned by the Government to grant leases on definite terms for a minimum of 50 years, these leases to have a lateral shore width sufficient for the most economical extraction of the mineral leased, taking into account the initial cost of the undertaking; the lessee to have undisturbed rights to the extension of the seams seawards between the parallels forming the shore boundary. A fixed rent would be agreed on, and in return for undisturbed seaward extension the operating company would guarantee to make roadways of an agreed permanent nature.

For a large modern mine designed for long seaward extension, a minimum of three miles of shore frontage would appear necessary.

THE COAL MINING INDUSTRY OF WESTERN CANADA†

(Jasper, Alta., Meeting, September 19th, 1927)

SECTION 1.—BRITISH COLUMBIA

By Robert Strachan (Member, C. Inst. M. & M.)*

INTRODUCTION

The coal mining industry of British Columbia is of comparatively recent origin, for it was only in the year 1835 that the first specimens of coal were brought to Dr. Tolmie, who was at that time attached to the Hudson's Bay Company's Fort McLoughlan post on Millbank sound. These specimens were brought in by Indians who were trading at the Post, and they naturally created much interest. They were reported to have come from the northern end of Vancouver island. The value of the find seems to have been very fully appreciated, and, in the following year, sufficient coal was collected to supply the Beaver, the first steamboat to ply on the Pacific coast, and also for blacksmithing purposes. However, the discovery of coal in British Columbia, even though it predates by twenty years the discovery of gold, did not create the excitement that the latter did, nor has its effect on the history of the Province been in any way comparable to that of gold.

That coal will play an important part—even more important than has gold—in the future development of the Province, is not to be doubted, when we consider the history

[†] Includes the following papers:

Section 1.—British Columbia, by Robert Strachan;

Section 2.—Alberta, by W. J. Dick;

Section 3.—Saskatchewan, by R. J. Lee;

Section 4.—Contributed remarks, "Outbursts of Gas and Coal in Coal Creek Colliery, B.C.", by Bernard Caufield.

^{*} Inspector of Mines, Fernie, B.C.

of the coal mining industry in other countries and especially in Great Gritain. In this connection I should like to repeat the words of the late Viscount Long, who presided over the first Empire Mining and Metallurgical Congress in London, in 1924:

"A little over a hundred years ago commenced in England an era of extraordinary progress, an industrial revolution, which has changed radically the whole national life and indirectly affected the whole course of civilization.

"This progress, this peaceful revolution, was founded upon the exploitation of the great resources of *coal* and *iron*. Without these resources, the England of today would not have come into existence; it would have remained a small country of eight to ten million people, largely agricultural. It could not have become, as it has, the source and mainspring of the industrial progress of the world.

"In the survey of the mineral resources of the Empire, it is only natural and proper that coal should come first. It is the basis, not only of industrial activity, but to a large extent of individual comfort and efficiency. Without coal we could not run our factories, heat our houses, or cook our food.

"Coal is the foundation of a whole series of industries, apart from immediate by-product coke, including ammoniacal liquors, tar dyes, oil and gas for heating and illumination. Coal brought, either directly or indirectly, the railway, the locomotive, and the steam engine to the service of man."

The important part played by coal in modern industrial development is well summed up in the following paragraph which appeared in the Vancouver *Province* on December 8th, 1924:

"Cheap coal is the rock on which Great Britain has built up her industrial and commercial prosperity. It has furnished the fuel for her smelters, the motive power for her factories, and, shipped in vast quantities to the Continent, has become the medium of exchange.

"It was the lure of coal that drew the German armies into Belgium and northern France, and it was coal, again, that took the French into the Ruhr.

"Coal means motive power, and motive power is the life-blood of industry."

It is not necessary to enlarge further on the unique part adequate supplies of coal have played in the industrial growth of all civilized nations. The Province of British Columbia is well endowed with such resources, and the coal-fields are so distributed as to be easily accessible to nearly every part of the Province. They are situated, broadly speaking, in three separate districts, in each of which a certain amount of development has taken place.

These districts may be distinguished as: (1) the Western, including Vancouver island and adjacent Gulf islands and Graham island; (2) the Central, in the interior of the Province, and including the Nicola, Similkameen, Skeena, Peace and Thompson River valleys; and (3) the Eastern, including the Crowsnest pass, the Elk and Flathead River valleys.

The economic development in each district has naturally followed the settlement of the Province, and the demands for fuel to provide heat, light, and power. Vancouver island being the first settled portion of the Province, naturally witnessed the earliest development, which was closely connected with the Hudson's Bay Company. As previously stated. this was in the year 1835, when some coal was produced from the northern end of the Island for the steamship Beaver and for blacksmithing purposes.

Production continued on a small scale, and in the years 1849-50 miners were brought out from Scotland with a view to determining the extent, regularity, and value of the coal measures. Among these miners was Robert Dunsmuir, who was destined to play a very important part in the development of the coal industry on Vancouver island.

Most of the work up till this time had been in the northern portion of the Island, but reports came from the Nanaimo district that coal existed there, and was being mined, having been discovered by Coal Tyhe, an Indian. James Douglas, Chief Factor for the Hudson's Bay Company, instructed

Joseph McKay, local factor, to take charge of the Nanaimo coal fields on behalf of the Company. His letter, dated 24th August, 1852, is still preserved, directing Mr. McKay to forbid the working of the coal, either directly by their own labour, or indirectly through Indians or other parties employed for that purpose, except under the authority of a license from the Hudson's Bay Company (Figure 1).

That the result of the work in the Nanaimo area was more encouraging than that in the northern part of the Island seemed assured, for before the end of the following year, 1853, over two thousand tons of coal had been produced.

From this small beginning, the coal industry in British Columbia has steadily grown, despite many obstacles, keeping pace with the demands of local as well as export trade, until, in the year 1925, the production reached 2,328,522 long tons, valued at \$11,642,610, and 5,443 workmen were employed in the mines. The total output to date has amounted to 65,926,604 long tons, valued at \$247,353,353.

The metalliferous mining industry in the Province owes much to the coal mines, which have provided an ample supply of coke, amounting to 4,393,255 tons valued at \$25,673,600. In this way they have assisted in retaining to the Province the smelting of a very large percentage of the minerals produced. It is generally conceded that the ample supply of cheap coke and coal, especially that from the Crowsnest coal field, saved the vast tonnage of the Rossland mines from being shipped to the United States to be smelted. It also proved an important factor in the decision to build smelters at Trail, Grand Forks, and Greenwood, with the result that the coveted copper tonnage of the Boundary district was retained in the Province.

Coal has been an important factor of exchange in international trade, for as early as the year 1895 we find that almost 80 per cent of the coal produced in the Province was exported. These exports were principally to San Francisco and San Diego, in California, but included also shipments to Cregon, Washington, Alaska, Petropavlaski, Hawaiian islands, and to Acapulco in Mexico. In 1925, about 10 per cent of the Vancouver Island output was exported, and

Fort Victoria 24 th Sugart 1832 Mi Claseph A Kay You will proceed with all possible diligence to Wentuhuysen Inlet Commonly Known as Nanyons Bay and formally take passession of the Call Beds later diservered there for and in behalf the Hudsons Bay Company Upon well give due notice of that proceeding to the Masters of all ressels arriving there, and you will forbid all persons to work the Coal either directly by means as their own labour or indirectly through Ordians or other parties employed for that purpose, except under the of a beende from the Hudson's Bay Company

Figure 1a.

You will require from Versons as may bo to work Coal by the Culson's Company, security a royally or will levy on the Spot, upon all fral whether procured by Mining in by purchase Natives the and from time to time to the ceanled On the event of any evasion of those regulations you will immediately takes to communicate intelligence of the same to mo.

Douglas

Figure 1b.

The above is a facsimile of a letter dated August 24th, 1852, from James Douglas, then Chief Factor of Hudson's Bay Company in British Columbia, to Joseph McKay, a Factor of the Company, to proceed to Nanymo (Nanaimo) Bay and to take possession "of the coal beds lately discovered there for and in behalf of the Hudson's Bay Company," and to levy a royalty on all coal mined there by any one. This is the first authentic record of the commercial production of coal in this area.

29.19 per cent of that from the Crowsnest Pass district, so that, from the latter district at least, the export trade is still of considerable importance. There is also a small export of coke, principally to points in the United States.

The investment in the coal mines of the Province is estimated at about thirty-three million dollars.

THE WESTERN DISTRICT

Historical

In this district, there has been production on a commercial scale from mines on Vancouver Island only, and the history of these mines is, to a great extent, the history of the coal mining industry of the Province.

From the year 1855, when the first attempt was made to mine coal at the Park Gate mine on Vancouver island, until 1898, when the Crow's Nest Pass Coal Company commenced operations in the Eastern district, the coal mining industry was confined to Vancouver island. Reference has already been made to the Hudson's Bay Company's operations in the vicinity of Nanaimo in 1852-53. The Company decided to bring out a number of English miners to work in the Nanaimo mines, and these, accompanied by their families. sailed from London dock on the 1st of June, 1854. In the light of modern means of transportation, when one can leave London and be in the city of Nanaimo two weeks later, it is interesting to read that these miners, travelling in the sailing ship *Princess Royal* via Cape Horn and calling at the Sandwich islands, did not arrive at Nanaimo until November 27th. one hundred and eighty days, or just half a year, after leaving London. The arrival of these miners no doubt gave an impetus to the coal mining operations, and in the following year a loading wharf was built from which small boats were supplied with coal. This wharf can be seen in Figure 3, reproduced from a photograph of Nanaimo harbour taken in 1858.

In 1861, the Hudson's Bay Company sold their coal interests in this area to the Vancouver Coal Mining and Land Company, who, with their successors, have produced over nineteen million tons of coal.

The following is a list of the mines opened in the Nanaimo coal field by the Hudson's Bay Company and their successors, who have been, in the order named, the Vancouver Coal Mining and Land Company (1862), the New Vancouver Coal Mining and Land Company (1881), the Western Fuel Company (1902), the Canadian Western Fuel Company (1918), and lastly the Western Fuel Corporation of Canada, Ltd. (1921):

Mine	Location	Year opened	Remarks
Park Gate Park Gate Newcastle No. 3 Pit Old Douglas Wellington Fitzwilliam Harewood New Douglas South Wellington East Wellington No. 1 Nanaimo Northfield Reserve Wakesiah	Nanaimo Brechin	1855 1856 1855 1857 1862 1870 1872 1874 1875 1878 1882 1881 1903 1910 1918	Abandoned "" "" "" Working Abandoned Working ""

Mr. Robert Dunsmuir, one of the first miners brought out from Scotland, worked, at first, for the Hudson's Bay Company and other interests, but in 1869 he commenced prospecting on his own account and having discovered some coal seams near Wellington, he organised the Wellington Collieries. He later developed the mines at Cumberland, where coal had been found at Baynes sound, proving the existence of a coal field in the Comox-Cumberland area. These mines were operated by the Union Coal Company. Dunsmuir also opened a mine at South Wellington in 1895 and the Extension mines in 1899, while the No. 5 at South Wellington was opened by the successors to his interests in 1918. These several properties, as well as a land grant in connection with the building of the Esquimalt and Nanaimo railway, a large portion of which is underlain by coal, made the Dunsmuir interests a large factor in the coal industry of Vancouver island.

In 1886, the New Vancouver Coal Mining and Land Company and the Dunsmuir interests together produced ninety-one per cent of all the coal mined on Vancouver island—and incidentally in the Province—while during the year 1925 the successors to these two companies, the Western Fuel Corporation of Canada and Canadian Collieries (Dunsmuir), Limited, produced seventy-five per cent of the coal mined on Vancouver island, and almost forty-four per cent of the entire production of the Province.

Geology of the Western District (1)

The Western District of British Columbia, as defined for the purpose of this article, includes the coal fields of Vancouver and Queen Charlotte islands. This region is nearly 600 miles long in a northwest direction, and is underlain mainly by Triassic and Jurassic sedimentary and volcanic strata invaded by batholithic granitic bodies of late Jurassic or early Cretaceous age. The sediments and volcanics have been folded and faulted. Following the period of mountain building, the region suffered pronounced erosion. Extensive areas with surfaces of moderate relief were developed and on these, in Upper Cretaceous time, was deposited a series of sediments, in places 10,000 feet or more thick. The Upper Cretaceous beds have since been much eroded and are now confined to a number of isolated areas which, for the most part, are low-lying and bordered by mountainous districts. The areas of Upper Cretaceous strata constitute the main coal fields of the Western District.

In Tertiary time, the coal-bearing Upper Cretaceous rocks were folded and tilted, but in most districts they still lie with low inclinations in gentle folds. Locally the structures

⁽¹⁾ The section on geology has been prepared by Dr. G. A. Young, of the Geological Survey of Canada.

are more pronounced. In the Tertiary era, igneous activity and sedimentation took place. In several localities, relatively small bodies of igneous rocks invaded the Upper Cretaceous beds and, in several districts, large volumes of Tertiary sediments formed and have been in part preserved. These Tertiary sediments in some districts, such as the Fraser River delta area, contain thin seams or lenses of lignite, but these so far as known, are not of commercial importance. On Graham island, one of the Queen Charlotte islands, Tertiary lignite occurs in seams of considerable thickness. This occurrence is referred to in connection with the description of the Cretaceous coals of Graham island.

The important areas of Upper Cretaceous strata are as follows: Nanaimo, Comox, Suquash; Gulf Islands; Cowichan; and Alberni. Coal-bearing Cretaceous strata also occur on Graham island and on Quatsino sound, but may there be of Lower Cretaceous age.

Nanaimo Area (1):

The Nanaimo Upper Cretaceous area includes the Nanaimo coal field, the most important in the Western District. The area is on the east coast of Vancouver island, almost directly west of the city of Vancouver. It extends from Nanoose bay southeast to the mouth of Ladysmith harbour and is continued southeastward by the Gulf Islands area. It has a length of 25 miles and, including Gabriola island, a maximum breadth of 17 miles. Its total area is 160 square miles, of which about 65 square miles are coalbearing. The area is bounded inland by high ridges of Triassic, and possibly Jurassic, volcanic and sedimentary strata invaded by granitic rocks. The Upper Cretaceous beds, known as the Nanaimo series, rest on an erosion surface of the older rocks possessing considerable relief, possibly as much as 2,000 feet.

The Nanaimo series consists of alternations of conglomerate, sandstone, and shale with an average total thickness of 6,760 feet and a maximum thickness of 9,000 feet. The strata have a general low dip to the northeast and lie in open

⁽¹⁾ Clapp, C. H.: Geol. Surv. of Canada, Memoir 51.

folds whose axis in most cases strike northwest. The beds of the limbs of the few larger folds dip, on an average, at angles of 10 to 20 degrees and as high as 50 degrees. Minor folds are numerous. Only four faults of any great size traverse the strata, but there are many minor faults.

The Nanaimo series, as shown by its fauna, is of Upper Cretaceous age and is in part of marine origin. But the series also contains plant remains as well as coal and, therefore, fresh or at least brackish water conditions probably alternated with marine conditions. The series has been divided into eleven formations each with distinguishing characters although consisting of rapid alternations of the same general rock types. The coal occurs in the lower part of the Nanaimo series in three productive seams, the Wellington, the Newcastle, and the Douglas. The lowest seam, the Wellington, lies about 700 feet above the base of the Nanaimo series; the Newcastle overlies the Wellington by 800 to 1,000 feet; and the Douglas is 25 to 100 feet higher. A small seam, called the Little Wellington and averaging about 2 feet thick, lies 20 to 60 feet above the Wellington in the northwest part of the productive area. Another small seam occurs further southeast 80 to 100 feet above the Wellington seam. In another locality, a seam with a maximum thickness of 2 or 3 feet lies 200 to 250 feet above the Wellington, and throughout the district there are numerous lenses of coal in the lower two-thirds of the Nanaimo series. Most of these lenses are only a few inches thick, but a few are 2 to 4 feet thick.

The three main seams "are remarkably persistent considering the great variability of the associated rocks, but vary greatly in thickness and quality. In places a variation as great as from 2 or 3 feet of dirty, slickensided coal, or 'rash', to 30 feet of clean coal occurs within a lateral distance of 100 feet. It seems as if this extreme variation is due to a folding of dirty or silty coal seams when at least the clean coal was in a plastic or pasty condition permitting it to flow away from the bends, where an increased vertical pressure was developed, to the limbs of the folds, where there was a corresponding decrease of pressure. There are also large, barren places in the seams owing to silting or similar cause. The coal seams are displaced also by small faults,

Seam:	Wellington		Newcastle		Douglas	
	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
Proximate analysis:						
Moisture	1.65	1.16	1.10	1.90	1.60	1.54
Vol. combustible	43.25	40.47	39.30	39.40	37.70	33.50
Fixed carbon	45.52	50.04	49.20	45.70	47.70	56.23
Ash	9.24	7.80	10.00	11.70	10.10	8.44
Sulphur	1.24	0.53	0.40	1.30	0.90	0.49
Fuel ratio	1.05	1.23	1.25	1.16	1.20	1.69
Ultimate analysis:						
Carbon	72.80	75.53	72.10	67.70	71.00	74.46
Hydrogen	5.17	5.13	4.7	4.7	4.9	5.42
Nitrogen	0.88	1.19	1.2	1.2	1.2	1.37
Oxygen	10.67	9.82	11.6	13.4	11.9	9.82
Sulphur	1.24	0.53	0.4	1.3	0.9	0.49
Ash	9.24	7.80	10.0	11.7	10.1	8.44

No. 1—'Run of mine' from No. 1 mine, East Wellington.

No. 2—'Run of mine' from Extension mines.

No. 3—Regular sample of commercial coal, over 1½-inch screen; Canadian Collieries.

No. 4—Regular sample of commercial coal, over 2-inch screen; Western Fuel Corporation, Nanaimo.

No. 5—Regular sample of commercial coal, over 2-inch screen; Western Fuel Corporation, Nanaimo.

No. 6—'Run of mine' from South Wellington mine of the Pacific Coast Coal Mines Co.

Representative analyses from each seam

Seam:	Douglas	Newcastle	Wellington	Comox	
Fixed carbon	48.5 41.2 10.3 2.2 1.18	46.6 41.5 11.9 2.4 1.12	49.8 40.1 10.1 1.8 1.24	57.8 30.2 12.0 1.0 1.91	
B.t.u.	12,830	12,470	13,160	13,010	

although an actual break seldom occurs, the coal having been forced along the plane or zone of dislocation. Rarely the seams fold or wrinkle without any appreciable variation in thickness'.

The Wellington seam is well developed in the northern part of the productive area, where it averages 3 to 6 feet in thickness and consists almost entirely of clean coal. Further south at one outcrop the seam attains a thickness of 10 feet. but the amount of clean coal in places is less than a foot. Still further south the average thickness is from 6 to 10 feet, but in many places the seam contains thick partings of carbonaceous shale and bony coal. The thickness where mined varies from almost nothing to nearly 30 feet and perhaps averages 4 to 7 feet. Other impurities besides partings and dirty broken coal are of little consequence. conspicuous feature of the seam is its variability in thickness, caused chiefly by minor faults, folds, or bends, [but] the floor is fairly regular and even... The coal of the Wellington seam where relatively undisturbed is black... In places it is finely laminated... and in other places it is massive... The coal is fairly hard and strong, breaks with a hackly fracture, and weathers well... As is shown by ... analyses... the Wellington coal is a rather high-volatile bituminous coal and does not differ essentially in composition from the other Nanaimo coals".

The Newcastle seam where mined has an average thickness of "from 30 to 40 inches with extremes of 20 inches and 6 or 8 feet... To judge from the few available analyses...the Newcastle coal, while not differing in any pronounced degree from the Wellington and Douglas coals, being a high-volatile bituminous coal, appears to be lower in fixed carbon and actual carbon content, and higher in oxygen and ash...".

The Douglas coal-seam probably averages about 5 feet in thickness, but it varies from almost nothing to over 30 feet. Pinches and swells are caused chiefly by irregularities in the floor. The coal is black with a sub-metallic to brilliant lustre. It is massive, fairly hard, and weathers well. It is a high-volatile, bituminous coal very similar to the Wellington coal.

The following estimate of the coal resources of the Nanaimo field was prepared by the late J. D. Mackenzie of the Geological Survey of Canada. In the table 'actual coal' means coal certainly known to be present; 'probable coal' means coal known to occur or inferred to occur, but the quantity of which has not been closely enough established to warrant classifying it as 'actual coal'; 'possible coal' means coal known to occur but only as isolated outcrops or under conditions that do not warrant its classification as 'probable coal'.

Estimate of Coal Resources, Nanaimo Area

		A	Below 2,000 feet		
		Actual	Probable	Possible	Possible
Douglas and	3 feet or more thick	27,160,000	59,770,000	68,000,000	35,000,000
Newcastle seams	1 foot to 3 feet thick	4,402,000	6,867,000		
Wellington seam	3 feet or more thick	6,910,000	16,450,000		
ccam	1 foot to 3 feet thick	3,400,000	13,975,000	54,000,000	
Grand total .		41,872,000	88,062,000	122,000,000	35,000,000

Comox Area(1):

The Comox Upper Cretaceous area borders the northeast coast of Vancouver island from Nanoose bay, a short distance north of the Nanaimo area, to Campbell river, a distance of about 75 miles in a northwest direction. The area occupied by the Upper Cretaceous beds is 420 square miles. The

⁽¹⁾ Mackenzie, J. D., Manuscript report, Geol. Surv. of Canada.

measures belong to the Nanaimo series and are equivalent in age to an upper part of this series as developed in the Nanaimo field. The strata border the coast and extend inland to the edges of ridges and mountain ranges of older rocks. Coal occurs in various districts and is being produced in the Cumberland field which, with the Tsable River field to the south, consists of 80 or 90 square miles lying in a strip 6 to 7 miles wide between Brown and Puntledge rivers on the north and Tsable river on the south.

"In this area the sedimentary Nanaimo series... rests unconformably on rocks... of Lower Jurassic and Triassic age... [consisting of] altered effusive and pyroclastic volcanics which were deeply weathered in pre-Upper Cretaceous time. On their weathered surface... was accumulated the Comox formation of the Nanaimo series, which is dominantly composed of massive, homogeneous, white and light-grey sandstone, containing coal seams. Locally at its base are found coarse angular breccias... and more rarely a basal conglomerate... Characteristically, however, the basal measures are fine; sandstone, shale, and even bony coal having been observed lying directly on the pre-Cretaceous volcanics. The Comox formation is of variable thickness, ranging from 80 to 1,000 feet and averaging about 600 feet; the variation being principally due to the inequalities of the surface on which it was accumulated".

The Comox is succeeded by alternations of shale, sandstone, and conglomerate, having in all a thickness of about 6,000 feet. These beds are the equivalents of the upper two-thirds of the Nanaimo series as developed in the Nanaimo area to the south. The Comox formation with its coal seams is equivalent to the formation which in the Nanaimo area lies immediately above the member holding the Douglas coal seam, the highest of the main coal seams of the Nanaimo area.

"The Comox formation underlies a strip of country from $1\frac{1}{2}$ to 2 miles wide extending with a southeast trend... Its southwest boundary where it is in contact with the volcanic rocks... is irregular; embayments of the Cretaceous extend back into the pre-Cretaceous rocks, and outliers of Comox sandstone are found lying on and surrounded by the older

metamorphic volcanics... [The strata of the Nanaimo series have a gentle northeast dip| modified in some places by folding and, also, by faults of small displacement. The average dip is about 5 degrees, but locally northeastward, inclinations up to 50 degrees have been observed and in some parts of the field low dips in the opposite direction occur... The distribution of the coal-bearing areas, however, seems to be only secondarily affected by the folding, and apparently the original inequalities of the pre-Cretaceous surface are of greater significance with respect to areas of workable coal than is post-Cretaceous folding".

"During the stage of the accumulation of the Comox formation there were recurrent episodes when coal seams formed. In the [southwestern part of the] area, five of these episodes may be clearly recognized by five usually quite distinct seams... In the vicinity of Cumberland the coal accumulation is not so clearly recognizable as belonging to distinct intervals, but even here there is some evidence that the thicker coal seams at any rate were accumulated at four or five separate horizons. It is also generally, though not always, true that the thicker seams were formed near the base of the measures, and that the seams formed later are thin and unworkable".

"Characteristically, the coal is associated with layers of shale ... there being nearly always more or less shale above or below the seam, and frequently the coal is wholly enclosed in shale. Like the seams in other parts of Vancouver island, these have no trace of anything resembling underclays, nor have rootlets, tree stems, branches, or leaves been observed in association with the coal itself... More or less fissile carbonaceous shale, and the brown compact shale known as 'bone' occurs interbedded with the coal itself. These impurities vary from a lamina of paper thinness, to bands occupying most of the thickness of the seam; and instances occur where the seam consists of shale, or of coal so high in sediment as to be unworkable. This is particularly the case where the seam closely approaches the pre-Cretaceous rocks... In one of the mines at Cumberland, working in a seam usually 100

feet above the base of the measures, the coal becomes thin, poor, and unworkable where it overlies one of the protuberant knobs of the old floor".

"Still another characteristic of some of the seams is the manner in which they split and reunite through the intercalation of lenticular layers of shale or sandstone... The thickness of coal in any given seam may vary from a fraction of an inch to many feet, 25 feet of coal being the thickest obtained in any given seam. This, however, included a band of shale 4 inches thick, and the coal was soft and shaly. solid bench of bright, hard, clean coal exceeding 30 inches in thickness is an unusual occurrence, though benches up to 35 inches are known. The upper of the two seams now being mined at Cumberland averages 3 feet 8 inches of coal: the lower one, about 100 feet below and near the base of the measures, averages 4 feet 2 inches".

Southeast of the Cumberland field, the Nanaimo series occupies about 120 square miles and extends to Nanoose bay. No coal is known to occur in this southern area. North of the Cumberland field, the Nanaimo series extends to Campbell river and lake and occupies an area of about 240 square miles. Immediately north of the Cumberland field. the Comox formation in the Dove Creek-Brown River field outcrops along the western edge of the area of the Nanaimo series, with a general low, northeasterly dip modified by some gentle folds. Coal occurs in several seams which range in thickness from 1 to 5 feet. Further north in the Tsolum River field, a seam 4 feet thick is exposed. Still further north in the vicinity of, and southeast of, Campbell lake, the Comox formation underlies a large area, probably in the form of a syncline. In this general locality a coal seam is known to occur in various localities and to have a thickness of 2 feet or less.

Suguash Area (1):

Upper Cretaceous strata occupy the northeast coast of Vancouver island for a length of 14 miles between Port McNeill and Beaver harbour. The beds extend west, inland, for

⁽¹⁾ Dawson, G. M., Geol. Surv. of Canada; Ann. Rept., Vol. II. Clapp, C. H., Geol. Surv. of Canada; Sum. Rept., 1911.

Estimate of Coal Resources, Comox Area

		Above 2,000 feet				
		Actual	Probable	Possible		
Cumberland field	3 feet or more thick	5,900,000	19,500,000 44,000,000	38,500,000		
Tsable River field	3 feet or more thick	13,000,000	10,000,000	50,900,000		
Dove Creek- Brown River field	3 feet or more thick			5,230,000 40,000,000		
Tsolum River field	3 feet or more thick			75,000,000		
Campbell River field	3 feet or more thick		4,680,000 800,000	41,700,000 45,000,000		
Quinsam field	3 feet or more thick		4,820,000 1,250,000	33,900,000 30,825,000		
Grand total		30,900,000	89,050,000	427,055,000		

about 5 miles, and underlie Malcolm and other smaller islands to the east. The strata presumably are equivalent to parts of the Nanaimo series of the Comox and Nanaimo fields to the southeast. "Several seams of coal occur in a formation consisting chiefly of ... sandstone resembling that of the ... [coal-bearing Comox formation of the Comox area]. Interbeds of shale... are, however, thicker and more numerous... The structure of the measures is very regular and appears to be, in general, a broad syncline... The dips are very low, less than 10 degrees, and although there are several local rolls there are no sharp ones. The measures are broken by a few normal faults of very small displacement. The coal seams are also very regular and do not pinch and swell, as do those of the Nanaimo and Comox basins. The known seams are, however, thin, and the seam mined at present (1911) contains a large number of very persistent partings of various kinds".

The seams formerly mined by the Hudson's Bay Company, where visible on the beach at Suquash... "are two in number, the upper being at least 1 foot, and in places probably 2 feet, in thickness. It is separated by about a foot of soft shale from a lower seam with a maximum thickness of about 6 inches".

Culf Islands Area (1):

The Gulf Islands area is a southeastward extension of the Nanaimo area. The islands lie off the southeast coast of Vancouver island, extending over a length of 40 miles and, with the included narrow waterways, cover about 200 square miles. The area "is underlain throughout by the Nanaimo series which, in places, has a total thickness of at least 10,000 feet. With the exception of Saltspring island, on which the basal and lower formations are exposed, these islands are formed of the upper formations of the Nanaimo series. The region has been strongly folded, and several pronounced anticlines and synclines traverse the area in a northwest-southeast direction. Dips are usually high, and frequently vertical or even overturned, so that the structure is the most complex of any of the areas of Upper Cretaceous rocks".

⁽¹⁾ Mackenzie, J. D., Geol. Surv. of Canada, Manuscript Report.

"Though the rocks are exceptionally well exposed... no coal seams more than a few inches thick have been seen, and from the nature of the rocks it is not likely that they occur". Coal seams have been reported from several localities but probably are of no commercial value. The coal-bearing horizons of the Nanaimo area are barren where they outcrop and nearly everywhere else are 2,000 feet or more below the surface.

Cowichan Area (1):

The Cowichan area of Upper Cretaceous strata lies towards the southeast end of Vancouver island. It extends westward from the Gulf Islands area with a maximum breadth of 10 miles. It has a length of more than 25 miles but in its western part divides into three, elongated, basin-like areas. The Upper Cretaceous strata "consist of conglomerates, sandstones, and shales, with, in places, thin, coaly streaks and lenses associated with carbonaceous shales and strata". The measures are equivalent to the Nanaimo series as developed in the Nanaimo area, but only the lower members are present. "The rocks of the Nanaimo series ... are fairly well exposed; but no thick nor extensive coal seams are known, although small, lens-like seams are exposed in [one district]... These lenses, however, are rarely more than a few inches thick, but beds of impure sandy and shaly coal, from 3 to 6 feet thick [do occur in one district]... The lithological character of the [equivalents of the coal-bearing formations of the Nanaimo field]... is notably different, and no indications of persistent coal seams occur".

Alberni Area (2):

The Alberni Upper Cretaceous area lies in the central part of Vancouver island, at the head of Alberni canal, a narrow fiord extending northeasterly more than halfway across Vancouver island from its west coast. A low-lying area of about 30 square miles at the head of Alberni canal, is underlain by Upper Cretaceous sediments which form an elongated syncline, complicated by one principal anticline

Clapp, C. H., Geol. Surv. of Canada; Memoir 96.
 Mackenzie, J. D., Geol. Surv. of Canada; Sum. Rept. 1912, Part A.

and other lesser folds and by some faulting. The strata consist of shale, sandstone, and conglomerate, with carbonaceous and coaly shale, and thin coal seams. "Up to the present (1922) neither the natural exposures nor the prospecting operations have afforded definite indications of workable coal seams in the Alberni area".

Quatsino Sound Area (1):

Quatsino sound, with its various arms, is a long body of water opening on the west coast of Vancouver island about 25 miles south of its northern extremity. Cretaceous strata occur along the coast of the sound in two general areas. One of these borders the West arm for several miles on both sides of Coal harbour; the other, on the north side of the main channel of the sound, extends east and northwest from Winter harbour. The strata, in part at least, are of Lower Cretaceous age, older than the Nanaimo series.

The Cretaceous strata of the Coal Harbour field consist of shale, sandstone, and conglomerates, dipping southwards at angles of 10 to 30 degrees. Thin seams of coal and coaly streaks occur at two general horizons. At the upper horizon, one seam outcrops or is known at several places. It varies from about 2 feet of coal of fair quality to several feet of shale with seams of coal up to 6 inches thick. The second coal-bearing zone is probably from 400 to 500 feet lower than the first. In this zone seams of varying quality occur with thicknesses ranging from a few inches to several feet. The rocks of the Cretaceous area about Winter harbour also consist of shale, sandstone, and conglomerate. Coal is known to occur in one locality where, in vertical strata, coal, shale, and coal with shale, forms a horizon 6 feet thick.

Graham Island Area (2):

Graham island is the largest of the Queen Charlotte group and, with the exception of North island, it is the most northerly. Late Lower Cretaceous, coal-bearing strata occur in several detached areas in the south of the island and extend southward across Skidegate inlet, to Moresby island.

⁽¹⁾ Dawson, G. M., Geol. Surv. of Canada, Ann. Rept., Vol. II. (2) Mackenzie, J. D., Geol. Surv. of Canada; Memoir 88.

The Cretaceous strata lie on an uneven surface of older, metamorphic and igneous rocks. They consist of three members, the oldest of which is coal-bearing and is composed of sandstones and shales in lenticular beds having a total thickness which varies from 2,000 to 5,500 feet. The coal-bearing member occurs in two synclinal basins. "The largest of these basins underlies Skidegate inlet, and extends northward on Graham island for about 9 miles... It is complicated by several smaller folds... and the folding is more severe in the western than in the eastern limb". Part of the west limb is overlain by Tertiary volcanics. The second coal-bearing area is a few miles further north, is about 6 miles long, and is "a basin-shaped syncline, warped into several open folds complicated by minor crumplings and some faulting".

The coal possibly occurs at one general horizon and may be mainly confined to one seam. The seam or seams, from place to place, vary greatly in thickness and character and are involved in minor as well as major folds and faults. On the western limb of the southern syncline, where the strata are invaded by Tertiary igneous rocks, the coal is of an anthracitic character in some places, whereas in others it has a coke-like appearance. In the eastern part of the southern basin, the coal seam where exposed is folded and faulted and has a maximum thickness of more than 8 feet, of which several layers, varying from 2 feet to nearly 4 feet, are coal of a low-grade, bituminous variety. In the northern basin, a seam is exposed that varies in thickness from 4 to 18 feet and in places contains 16 feet of coal. The lower part is dirty, inferior coal, but the upper part is of better quality.

The eastern part of Graham island is presumed to be underlain, for the most part, by Tertiary sediments consisting of clays, sands, and gravels, and more indurated forms of the same sediments. The beds have a general low northeasterly dip, but in places have suffered folding and faulting. The total thickness is more than 1,000 feet; the age may be Miocene. Lignite is of widespread occurrence. At one locality on the north shore more than ten seams are exposed. They range in thickness from one to 15 feet and are of a tough, woody lignite.

Mining Operations in the Western District

Although the Nanaimo series contains three productive coal seams—the Douglas, the Newcastle, and the Wellington (and Little Wellington)—at no mine is more than one of these seams worked, except at Nanaimo colliery No. 1 mine, where, in addition to the Douglas seam, the Newcastle seam is also mined in one part of the mine. This seam is not found in any of the other mines. The Reserve, Cassidy, No. 5 South Wellington, and Morden collieries all mine the Douglas seam, and the Wakesiah, Extension, Lantzville, and East Wellington collieries the Wellington seam. The Old Wellington is the only colliery winning coal from the Little Wellington seam.

As mining and other operations at all the mines in the district are very similar, it will be necessary to describe only one mine at length, and for this purpose No. 1 mine at Nanaimo has been selected as typical of the district.

Nanaimo Area:

No. 1 mine.—This mine is owned by the Western Fuel Corporation of Canada. It is situated on the water's edge close to the city of Nanaimo and is probably the oldest working mine in the Province, having been in operation for almost half a century.



Figure 4.—No. 1 Mine, Nanaimo, from the sea.

The mine has four shafts, but only one of these, No. 1, is used for hoisting coal. This has been sunk 750 feet to the Douglas seam, and is a circular shaft, lined with concrete from surface to bedrock, and timbered from there down. Two cars, each carrying about 0.75 tons of coal, are hoisted at one time, the total weight of cars, cage, and coal being about 5 tons. The other shafts are used for ventilation, for taking in supplies, and for ingress and egress of miners.

The Douglas seam is mined on the south side of the mine, and its thickness is very variable, from three feet to about seven feet. The Newcastle seam is mined on the north side. It is about three feet thick, and is the only seam in the Nanaimo area that has any approach to regularity in thickness.

The mining methods have necessarily been adapted to meet the great irregularity in thickness of the Douglas seam. Where there is sufficient thickness, the pillar-and-stall method is employed, elsewhere longwall. Similarly, where the coal is thick it is mined by hand picks, but generally, coal-cutting machines, driven by compressed air, are used. These machines are of either the longwall or the post-puncher type—the former where comparatively long stretches of working face can be obtained, the latter where there are only short stretches.

The coal is undermined from four to six feet, and then brought down. In some cases it is necessary to use explosives for this purpose, and these must comply with the regulations in the Coal Mines Regulation Act, and the blasting be done with electric detonators and under the supervision of a competent person.

For the most part, the coal is loaded into the mine cars by hand, either by the miners or by special loaders, but in some cases face conveyors are used. In the longwall workings, roadways are furnished every 36 feet, but where the conveyors are used the roadways are as much as 200 feet apart.

Horses are generally used in hauling the loaded cars from the working places to partings or landings, where they are gathered into sets, or 'trips', of cars. There, according as the landing is in a slope or an incline, the trips are raised by compressed air hoists, or lowered by gravity planes, to the level roads, along which they are hauled to the shaft by electrical locomotives, either of the trolley or storagebattery type.

The regulations governing electric haulage underground in coal mines of the Province, as amended in 1925, state that: "Haulage by storage-battery locomotives may be used in any mine with the consent in writing first obtained of the Minister of Mines in all cases, and subject to such conditions affecting safety as may be prescribed by him. Haulage by electric locomotives of the overhead trolley-wire system is prohibited underground in any coal mine". In the case of coal mines where trolley systems were already installed at the time the amendments were passed, the new regulations will not come into effect until April 1st, 1930.

In common with all bituminous coal mines, there is, in the mines of the Nanaimo area, a certain amount of methane (fire-damp) given off by the coal, and to maintain the mines free from this and other deleterious gases emanating from workmen, horses, decaying timbers, etc., powerful ventilating fans are installed. In this mine there are two, one for the south and the other for the north side of the mine. Both are of the Sirocco type, steam driven, and together they deliver 270,000 cubic feet of air per minute, with a four-inch water gauge. This ventilation is divided into separate splits, as required by the Coal Mines Regulation Act, which provides that "Every mine shall be divided into districts, or splits, of not more than seventy men in each district, and each district shall be supplied with a separate current of fresh air."

Timbering at the working face is generally attended to by the miner, the timber being provided by the Company. Either posts with caps, or head boards, or cross-bars with a post at each end, are used. On main roadways this type of work is done by special workmen detailed for the purpose, and generally consists of heavier timbering, or, instead, concrete or brick, with or without steel cross-bars.

As the mine covers a fairly large area and the greater part of the workings are submarine, it makes a certain amount of water. This is dealt with by compressed air driven pumps in the in-bye workings, and by electrically driven pumps in other parts of the mine, which deliver the water to the shaft, where a storage is situated. From this storage large pumps, driven either by steam or electric power, raise it direct to the surface.

The Coal Mines Regulation Act prohibits the use of any lamp or light underground in any coal mine, other than a locked safety lamp of a pattern approved by the Minister of Mines. In these mines, the workmen underground use electric cap lamps, of either the Edison or Wheat type, and the officials generally carry in addition a Wolf safety lamp. This latter is for the purpose of making a rapid determination of the condition of the mine air with respect to the presence of methane, or fire-damp. An additional means of making this determination with greater accuracy is required by the Coal Mines Regulation Act, and for this purpose the Burrell Gas Detector is used.

The miners producing coal are paid so much per long ton for the coal mined and loaded, each miner having an identification tag, which he attaches to his car. Timbering and other work, such as ripping floor or roof, is paid for separately. The rate of pay is determined mutually by a committee representing the workmen and the officials of the colliery. It generally covers a period of years, and is renewable previous to the expiration of the agreement.

The surface plant at this colliery includes the usual offices, warehouses, machine and carpenter shops, power plant, with boilers and machinery, and the necessary plant

for cleaning the coal prior to shipment.

The power plant at this and other collieries in this field has been added to from time to time, growing with the development of the colliery and with advances in mining practice, and it frequently includes more than one type of boiler and even different types of similar machines. This condition very often arises when an industry has covered a long period of years, and frequently, when new and additional machinery is required, although it is a distinct improvement on the old, it its not always possible, or even economical, to entirely discard the old.

All the machinery at the Nanaimo colliery is driven by steam power. The present installation of boilers comprises two Babcock and Wilcox water tube boilers rated at 530 h.p.



each, and two 208 h.p. return tubular boilers. The two former have been installed recently and replace eight return tubular boilers, and, with the addition of chain grate stokers and super-heaters, they have effected a great economy both in the amount of fuel used and in labour. As the two new boilers produce steam at a higher pressure than the old type, a Locke pressure-reducer regulator is used to equalise the pressure to that in the latter. This boiler installation supplies steam to the machinery in the power house, two crosscompound compressors and two steam engines, which drive the direct-current electric generators, which supply the power for the electrical motors used both underground and on the surface, and for lighting purposes. The electric current is carried down the shaft by means of four armoured cables, and is distributed from a sub-station to points where needed for hoisting, pumping and lighting.

As the loaded mine cars reach the surface, they pass to a rotary gravity dump, capable of holding three cars at one time. On entering this, the car is rotated 120 degrees, thus allowing of the coal being delivered to the screen with a minimum of breakage.

The coal is separated into lump and slack, or small, the lump passing to the picking tables, where all foreign matter is taken out, and then to the railway cars for shipment. The slack is taken by belt conveyor to the washer, where it is sized and washed in washers of the Robinson type previous to being shipped. The cleaning plant at the Nanaimo colliery is also used for treating the smaller sizes of coal from the Company's other collieries, the Reserve and Wakesiah.

A large amount of the coal from this colliery is used either for bunkering ships or for shipment to the mainland, and large storage bunkers are maintained near Nanaimo harbour. Railway cars with a capacity of about six tons maintain a service between the mine and these bunkers, but when the coal is to be shipped to points on the Island the standard railway cars are used.



The majority of the workmen at the Nanaimo colliery reside in the city of Nanaimo, and many of them own their own homes, this city having the distinction of having more privately owned residences than any other city in the Province of British Columbia.

The Western Fuel Corporation does not maintain any change or wash rooms for the workmen at any of its collieries.

The Reserve colliery of the same Company is situated about five miles from Nanaimo, between which city and the colliery the Company maintains a passenger train service for its employees.

Here two shafts have been sunk to the Douglas seam, at a depth of 955 feet, where it is dipping at an angle of almost 15 degrees. Where intersected by the shaft, the seam is much disturbed, and it was deemed advisable to drive a horizontal rock tunnel from the shaft to the seam. In fact, owing to faults, abnormal thickness of coal, etc., the mining of the Douglas seam has probably been attended with more difficulty than in any other colliery. The coal is more friable than that in the Nanaimo mine, and is mined by hand, the pillar-and-stall system of working being adopted.

The Wakesiah colliery, also owned by the Western Fuel Corporation, is situated about two miles from Nanaimo, and here two shafts have been sunk to the Wellington seam, which is encountered at 320 feet.

At both these mines the methods of mining, haulage, pumping, lighting, etc., are similar to those at the Nanaimo mine, and all the machinery is driven by steam power.

Cassidy Colliery.—This mine, owned by the Granby Consolidated Mining, Smelting and Power Company, is situated about eight miles from Nanaimo, and was developed for the purpose of ensuring a steady supply of coke for their smelter at Anyox. The slack coal produced at the mine is shipped by boat to Anyox, where it is treated in by-product ovens to produce the coke necessary for the operation of the smelter. The lump coal finds a ready local and export market.

Three slopes have been sunk on the Douglas seam, which averages about ten feet in thickness at the outcrop, and dips at an angle of about 16 degrees. The irregularity in





Figure 7.—Cassidy Colliery, near Nanaimo.

thickness characteristic of this seam developed as the workings progressed, and, in addition, trouble has been experienced with severe outbursts of gas, principally methane, accompanied by the dislodgment of large masses of solid coal or by the discharge of fine coal dust. Several lives have been lost through this cause, and it has also tended to restrict the operation of the mine.

The coal is mined by the pillar-and-stall method, and is broken down by hand, except in a few cases where explosives Storage-battery electric locomotives are used are used. underground to haul the loaded cars to the main slope, from where they are taken to the mine tipple by a steam hoist situated outside the mine. The tipple and washer are modern in every respect, being equipped with rotary dump, Marcus screens, and loading booms. The washer equipment includes two-compartment jigs of 80 tons per hour capacity, seven Deister-Overstrom tables, and a 75-foot Dorr thickener. The steam plant consists of two Badenhausen boilers, and one Babcock and Wilcox boiler, all fired by mechanical stokers.

This colliery was opened in 1918, and the accommodation for the workmen is on a very elaborate scale. This begins at the mine, where a change room and wash-house is provided, with shower baths and lockers, and a special room for drying wet clothes. The whole building is well looked after.

On a bench overlooking Haslam creek to the south, and Nanaimo river to the north, an area of about 80 acres has been reserved and here a townsite has been laid out, with fine boulevards lined with trees. Good taste has been displayed in the style of architecture adopted for the houses, combining both convenience and a pleasing appearance, which is enhanced by the green lawns surrounding them. The town is furnished with modern water-works and sewage disposal systems. Recreation is provided for by a very fine athletic park containing baseball and football pitches, tennis courts, bowling green, and a quarter-mile track; and for general entertainment and physical and mental relaxation there is a community house, containing gymnasium, dance hall, library, and reading, billiard, and pool rooms.

For the accommodation of single employees, there is a large rooming house, containing 80 rooms, each with running hot and cold water. The rooms are completely furnished and attended to by the Company. The mess-house or dining room is in a separate building and is equipped with every modern convenience for ensuring labour saving as well as health conditions. The men enter first a long lobby, provided with an ample supply of wash-hand basins, and then pass through a vine-covered pergola to the dining hall, a bright, comfortable room, cool in summer and warm in winter, furnished with tables to accommodate groups of six.

Lantzville Colliery.—This colliery, owned by the Nanoose-Wellington Collieries, Limited, is situated about nine miles from Nanaimo. The Wellington seam is mined here, and is reached by two shafts at a depth of 133 feet. All the usual characteristics of this seam, as already described, are in evidence here.

Working conditions are very similar to those in the other collieries, the coal being cleaned and shipped to the mainland on scows from the Company's wharf close to the mine.

A considerable number of the employees reside in houses maintained by the Company near the mine, while others live in Nanaimo and travel to their work by car, jitney, or other means.

Wellington-Extension Colliery.—This colliery, consisting of four mines (Nos. 1, 2, 3, and 6) at Extension and one mine (No. 5) at South Wellington, is owned and operated by the Canadian Collieries (Dunsmuir), Limited.

The Extension mines are situated about seven miles from Nanaimo, and all are working the Wellington seam. Mines Nos. 1, 2, and 3 are connected underground by a cross-cut tunnel, while No. 6 is situated about 1¼ miles to the northwest of the main tipple. The coal from No. 6 is transported in cars over a narrow gauge railway about a mile in length, and is then dropped down an incline about 3,000 feet long to the main tipple.

These mines were opened by the Dunsmuir interests, and have continued producing coal ever since that time. The conditions underground with respect to mining are similar to those in the other portions of the field already described.

The South Wellington mine is situated about five miles from Nanaimo, and here the Douglas seam is mined. Two slopes have been driven from the outcrop a distance of 3,000 feet, and the seam has been found very much disturbed with faults and 'wants'.

At all the mines connected with the Wellington-Extension colliery, the coal is screened at the mine, and the lump coal is loaded separately, the slack coal or undersize being sent to the cleaning plant at Ladysmith. This is the shipping point for all the coal from this colliery and is connected to the mines at Extension by the Wellington Colliery railway, about eleven miles in length. The slack coal from South Wellington is shipped to Ladysmith over the Esquimalt and Nanaimo railway, a distance of nine miles.

The cleaning plant at Ladysmith is equipped with three washers of the Robinson type, each having a capacity of 200 tons in twelve hours, six compartment jigs and four 14 ft. by 7 ft. Mascoe tables taking care of the smaller sizes

of coal. Power for the cleaning plant is supplied by a Pelton wheel, and a 40 k.w. 240-volt Allis-Chalmers Bullock generator supplies power for lighting.

At Ladysmith the picked coal from the mines and the washed coal from the cleaning plant is either loaded on ships or in box cars, which latter are transported on barges to mainland points.

The majority of the employees in the Extension colliery reside in Ladysmith and reach their work by passenger train run over the Wellington Colliery railway.

The two other collieries mentioned as operating in this district are those of Island Collieries, Limited, and of the East Wellington Coal Company. Both are small and are mining very thin isolated portions of the Wellington seam.

At the Island, or Old Wellington, colliery (better known as King and Foster's), the seam being mined is the Little Wellington, and this is the only place where this seam has proved economically mineable. The mine is in some old workings abandoned by the Wellington Colliery over twenty-five years ago.

At East Wellington the operations consist of two longwall faces, 550 feet and 300 feet, respectively, where the coal is mined by coal-cutting machines.

Morden Colliery.—This colliery, owned by the Pacific Coal Company, is situated about 6 miles from Nanaimo, and the seam mined is the Douglas. Two shafts reach the coal at a depth of about 650 feet, and a very fine reinforced concrete head-frame has been erected. No coal has been produced from this mine for several years, owing to the Company being in liquidation. When operating, the coal is transported over the Company's own railway to Boat harbour for shipment.

The same Company also owns a property at Suquash, twelve miles north of Alert bay, but this has been closed down for some years. The seam at Suquash averages about 6 feet in thickness, but no attempt has been made to correlate it with the seams at Nanaimo or Comox. While a limited

amount of plant has been installed at Suquash, it will be necessary to construct a railway to tide-water before coal can be shipped. As very little mining has been done here, it remains yet to determine the best methods of mining and other details of operation.

Comox Area:

Comox colliery.—This colliery, the only one operating in the Comox area, is owned and operated by the Canadian Collieries (Dunsmuir) Ltd. and is situated about 70 miles north of Nanaimo and about six miles in a westerly direction from Union bay, which is the main shipping point. While eight mines have been opened in this district, at the present time only three are active. Two of these are producing coal, the other being kept for ventilation and drainage.

There are ten coal seams in this area, and they possess the same characteristics as the seams in the Nanaimo district, with pinches, rolls, and swellings, but to a lesser extent; 'wants', however, due to silting, are even more frequent than in that district. Of the ten seams found, only three have been deemed of sufficient importance to warrant mining operations. These are known as Nos. 1, 2, and 4.

The thickness of the seams varies from three feet to over eight feet. From the accompanying logs of bore-holes (¹) through seams Nos. 1-4, it will be readily seen that the mining of these seams is a difficult and expensive undertaking. It is impossible to foretell what the conditions may be even 100 feet from any point, and as a result planning work ahead is practically out of the question.

No. 4 mine, the largest producer, is entered by two slopes, the main haulage slope having been driven for a distance of 9,000 feet on the coal seam, while a branch off this slope has penetrated a distance of 7,000 feet. The seam worked is No. 4, and it varies in thickness from six feet to fifteen feet, which includes several bands of shale.

⁽¹⁾ Reproduced from paper by Charles Graham on "The Problems of the Vancouver Island Coal Industry"; Trans. Can. Inst. Min. & Met., Vol. XXVII, 1924, p. 466.

Coal seams in the Comox district

Sections of each seam in bore-holes 163, 22, and 166

No. 1 Seam

Bore-hole 163		Bore-hole 22		Bore-hole 166
Ft. Coal. 0 Shale. 0 Coal. 0 Shale. 4 Coal. 1 Shale. 5 Coal. 1 Shale. 1 Coal. 1 Coal. 0	in. 4 11 1 4 0 9 5 8 6	Ft. Coal. 0 Shale. 3 Coal. 0 Shale. 1 Coal. 1 Shale. 0 Coal. 0 Shale. 1 Coal. 0 Shale. 1 Coal. 0	in. 8 0 1 0 10 10 8 0 8	Ft. in. Coal. 1 0 Shale. 1 6 Coal (dirty) 1 0 Shale. 0 8 Coal (dirty). 1 0 Shale. 2 0 Coal. 1 4
		No. 2 Seam		
Coal. 2 Shale. 0 Coal. 2 Shale. 0 Coal. 0 Shale. 0 Coal. 1	10 1 2 6 4 7 5	Coal 4 Shale 0 Coal 0 Coal and shale 0 Coal 1	6 7 5 11 4	Coal 1 10 Bone 0 2 Coal 1 2 Shale 3 6 Coal (bony) 2 0
		No. 3 Seam		
Coal. 1 Shale. 0 Coal. 1 Shale. 1 Bone. 0	4 2 6 10 2	Coal and shale mixed, 1 Coal 1	8	
		No. 4 Seam		
Coal. 0 Coal, bony. 1 Shale. 2 Coal. 1 Bony coal and shale. 0 Bony coal and shale. 1 Dirty coal. 1 Shale. 5 Coal, dirty. 3 Shale. 7 Bony coal. 0	6 0 0 6 6 6 6 8 4 0 0 6	Coal and shale 0 Coal	6 4 5 5 6 9 5 5 9 2 2	Bony coal

No. 5 mine is a vertical shaft, which reaches the No. 1 seam at a depth of 280 feet, and a short distance from the shaft a slope has been sunk to the No. 2 seam. No. 1 seam varies in thickness from three feet to four feet, and No. 2 from five feet to seven feet, both including several bands of shale.

The method of working is either pillar-and-stall or longwall, depending on local conditions. The coal is machine-mined as a rule, but under exceptional conditions hand picks are used. Electric power is employed for the coal cutting machines and for other purposes, such as haulage and pumping underground, to a greater extent than in the Nanaimo district. Ventilation is produced by electrically driven fans of large capacity, either of the Sirocco or Sullivan type.

Power is obtained from the Company's own hydro-electric station situated on Puntledge river. The plant here consists of two Francis turbines, each 6,000 h.p. capacity, direct connected to two 4,000 k.w., 13,200-volt, three-phase, 25-cycle generators. The erection of this plant was commenced in 1912 and power was being delivered the following year, thirty miles of transmission line distributing it to the colliery and surrounding towns.

The coal is screened at the mine, and then shipped either to the railway (which connects with the Esquimalt and Nanaimo railway at Royston) for distribution, or to the Company's shipping point at Union Bay. All the fine coal is sent direct to Union Bay over the Company's Wellington Colliery railway, and there it is sized and washed previous to being shipped.

The coal from the Comox mines is generally regarded as a high-carbon bituminous coal, suitable for coking, and early in the development of the industry (in 1896) coke ovens of the bee-hive type were erected at Union Bay, convenient to the washing and screening plant. This was the first coke-making plant in the Province, and it continued in operation for many years—continuously from 1897 to 1910 and again from 1915 to 1919, during which time labour troubles caused a stoppage of coke supplies from the Crowsnest coalfield. The establishment of coke ovens in the latter coalfield was no doubt largely responsible for the closing down of the Union Bay plant.

The Crowsnest not only has the advantage of a shorter haul to the smelters, but a better quality coke is produced from the coals in that field. However, there were other factors. Thus, at the Comox plant, 573,462 tons of slack coal were made into coke, yielding 344,077 tons. In other words, it takes 1.66 tons of coal to produce one ton of coke, and if we take the average price of the coal used for coke-making, and the market price for the coke produced, we find that there was very little if any profit in the manufacture of coke at Comox.

573,462 tons of coal at \$3.75 per ton		,150,482 ,002,850
	_	
Loss	\$	147,632
or practically 42.2 cents per ton of coke produced.		

These figures would indicate that the only advantage gained in making coke was to absorb the slack or fine coal output of the mines.

General:

In the development of the Vancouver Island coal field, it will be noted that no effort has been spared to obtain the greatest possible tonnage of coal under very difficult conditions, and to place it on the market as advantageously as possible. At practically all the large collieries, cleaning plants, which include washing plants, have been installed with a view to eliminating foreign matter from the coal, and in doing so the average loss has been about 10.5 per cent, which, it should be remembered, has all been paid for to the workmen as coal.

The development of the Vancouver Island coalfield has been steady but it has not been commensurate with the development of other industries or of the general trade of the Pacific Coast; in other words, it has not made the progress we should have expected. This, to some extent, has been due to the troubles incidental to the working of the coal seams, but probably the greatest drawback in recent years has been the increasing use of fuel oil along the Pacific Coast. Not only has this cut off a large part of the foreign market for coal, but it has cut into the home consumption of coal.

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Large quantities of fuel oil, imported principally from the United States and Mexico, are now used in the Province on railways, in industrial plants, and even for heating purposes, and at a very conservative estimate this fuel oil takes the place of about one million tons of coal per year, the greater part of which would naturally be supplied by Vancouver Island mines.

Details of production from Vancouver Island collieries during 1925 are given in the table below:

Company	Colliery	Production (long tons)		Production per employee per day
Western Fuel Corporation of Canada.	Nanaimo	314,154 161,118	705 345	2.55
	Wakesiah	93,788	228	2.18
Canadian Collieries (Dunsmuir), Ltd.	Comox Extension	240,830 212,308	713 596	1.53 1.72
Granby Cons. M. S. &	S. Wellington	51,072	147	1.61
P. Co	Cassidy	196,079	275	3.15
CollieriesEast Wellington Coal	Lantzville E. Wellington	75,680 54,358	133	1.38
Island Colliery Co	Old Wellington	16,648	79	0.80
Total		1,416,035	3,462	

THE EASTERN DISTRICT

Historical and General

The Eastern coal mining district of British Columbia, generally termed the Crowsnest Pass coal field, lies on the western slope of the Rocky mountains in the extreme southeast of the Province.

The first mention we have of the existence of coal in this district is by a Mr. Blackstock who, in the year 1858, reported outcroppings of coal on the banks of the various creeks and rivers. In 1873, Michael Phillips, a trapper, noted the presence of coal in the Crowsnest pass and the following year he spent part of his time in prospecting the creeks in the vicinity. He reported the occurrence to the local government office at Fort Steele, at that time the principal trading post in East Kootenay, but little if any attention was paid to the discovery as there was no railway or other easy means of transportation to the district.

It was not until 1887 that Mr. William Fernie and Lieut.-Col. Baker commenced to systematically prospect and explore the district. They continued their work for eight years, and during that time were fortunate in interesting others to finance them. They then secured a charter to build a railway, which was accompanied by a grant of certain lands along the side of the railway, and included the rights to the coal.

The charter required them to build a railway from some point at or about the junction of Summit and Michel creeks to a point on the lower Kootenay river where it joined Goat river. This railway, known as the British Columbia Southern railway, forms the nucleus of the present Crowsnest Pass branch of the Canadian Pacific railway, and the land which accompanied the charter, with certain modifications, constitutes the holdings of the present Crow's Nest Pass Coal Company, comprising about 200,000 acres.

The Crow's Nest Pass Coal Company became actively interested in this field by acquiring a majority interest in the Fernie-Baker syndicate in the year 1897, and since that time they have played a leading part in the development of the coal industry in eastern British Columbia. Realising that a rail-

way, as laid down in the charter, provided only one outlet for the coal, namely over the Kootenay river to the Boundary district, the Company at once made an arrangement with the Canadian Pacific Railway Company to extend their branch line, then terminating at Fort McLeod, in Alberta, to the Kootenay river, or to Kootenay Landing on Kootenay lake as it is now termed, in order to enable them to open coal mines at Coal Creek and to build coke ovens at Fernie. The result was that in the following year, when the railway reached Fernie and a branch line extended to the mines, production immediately commenced.

During that year, 10,000 tons of coal were shipped, and also 361 tons of coke from the coke ovens, which were of the bee-hive type. The following year, coal production had increased to 102,000 tons and over 29,000 tons of coke were made. Since that time the Crow's Nest Pass Coal Company has been a continuous producer of coal and coke, and until 1908 they had the field to themselves.

In that year, two other companies started operations. The Canadian Pacific Railway, who under their arrangement to build the railway had acquired a block of six square miles, opened a mine at Hosmer, eight miles north of Fernie; and the Corbin Coal and Coke Company commenced operations on an outlying portion of the field, about twenty-four miles east of Fernie. The importance of the district in connection with the development of the interior of the Province will be realized when it is stated that, since the commencement of operations in 1898, over nineteen million tons of coal, or 27 per cent of the total production of the Province, has come from this district.

In the manufacture of coke, the Crowsnest coalfield soon outstripped the Western District. No doubt the shorter haul from the mines to the smelters exerted a considerable influence in this matter, but the superior quality of the coke also played a part. In this connection, it may be mentioned that, while coke ovens were erected on Vancouver island in the year 1896, or two years prior to the opening of the Crowsnest coalfield, almost 91 per cent of the coke made in the Province has come from the latter district.

As stated on an earlier page, it takes 1.66 tons of Vancouver Island coal to produce one ton of coke, whereas in this district it only takes 1.43 tons. If we assume in each case that the price of the coal used for coking is \$3.75 per ton and the price obtained for a ton of coke \$5.85, then coke was made on Vancouver island at a loss of 40.2 cents per ton, whereas in the Crowsnest it showed a profit of 49.2 cents per ton.

Since the district was opened, 19,634,386 tons of coal have been produced, of which 5,784,540 tons have been used in making coke, with a yield of 4,049,187 tons. At \$3.75 per ton, this coal, less the tonnage coked, would have a value of \$51,936,922, and the output of coke, at \$5.85 per ton, would be worth \$23,687,691; a total value of \$75,624,613.

The Crow's Nest Pass Coal Company has always been by far the largest producer in the district. Mines of this Company contributed 92 per cent of the coal produced in eastern British Columbia during 1925, and they have accounted for over 88 per cent of all the coal produced in the district.

Like other coal fields, the Crowsnest has had its troubles, and the principal one has been the lack of markets, owing to its being situated such a long distance from large cities or manufacturing centres, and to competition from coal mines in other districts, principally in the adjoining Province of Alberta. Another serious blow was the closing down of many smelters in the West Kootenay and Boundary districts, formerly large consumers of coke.

The dependence of this field on export trade is greater today than at any time in its history. In 1904, 17 per cent of the total coal produced was exported, whereas in 1922 exports accounted for almost 60 per cent of the output.

In 1904, 350,900 tons of coal, or 54 per cent of the total production, was made into coke, whereas only 61,497 tons, or 11 per cent, of the coal was made into coke in 1922.

The disposition of the output, taken over a period of twenty-two years, 1900 to 1922, has been as follows:

In recent years, the introduction of the Fordney tariff by the United States, which imposes a tax on articles entering that country when a tax is imposed on the same articles imported from the United States, has had some influence on the shipments. As coal imported into eastern Canada from the United States has to pay a tax of 53 cents per ton, coal imported from British Columbia into the United States has to pay a similar tax.

The peak production in this district was in the years 1910 to 1914, for which the figures follow:

Year	Production	Percentage of total production of the Province
1910	1,365,119	43.0
1912	1,261,212	43.6
1913	1,331,725	51.8
1914	955,183	44.0

The falling off in 1914 was due largely to the closing down in that year of the Canadian Pacific Railway Company's mines at Hosmer, which had been opened in 1908. Labour troubles, and the closing down of the smelters in the Boundary district, have contributed to reduce the output in subsequent years. For each of the past two years, 1925 and 1926, production has amounted to about 850,000 tons. The output of coke had steadily risen from 361 tons in 1898 to 286,000 tons in 1913. In 1925 not more than 75,000 tons were made.

Geology of the Eastern District (1)

The Eastern District of British Columbia, as defined for the purpose of this article, includes the coal fields of the southeast part of the Province. These coal fields lie within the Rocky mountains west of the crest line marking the boundary between British Columbia and Alberta.

⁽¹⁾ Prepared by Dr. G. A. Young.

The Rocky mountains continue uninterruptedly from the International Boundary to Liard river, a distance of 850 miles in a northwest direction. They are formed, almost entirely, of folded and faulted sedimentary strata that range in age from pre-Cambrian to late Mesozoic and succeed one another with conformable or nearly conformable attitudes. During most of Palæozoic and much of Mesozoic time, the site of the Rocky mountains was undisturbed by violent earth movements. It was part of a region that, during Palæozoic and earlier Mesozoic time, was occupied by successive seas in which great thicknesses of marine strata were deposited. Commencing in Lower Cretaceous time, the seas were more restricted and periods of marine invasion and sedimentation alternated with others of non-marine conditions during which great thicknesses of sediments accumulated, in some instances accompanied by coal. Later, about the opening of the Tertiary era, the Rocky Mountain region was involved in great earth movements and the mountains were produced. Within southeast British Columbia, the distribution of the strata, as the result of the structures and erosion, is such that, except in one area, coal is confined to one formation, namely, the Kootenay of Lower Cretaceous age. The one exception is a lignite-bearing Tertiary formation known in one area only. The coal-bearing areas are collectively known as the Crowsnest Coal Field.

Crowsnest Coal Field (1):

Mesozoic strata of southeastern British Columbia, occur in a number of detached areas distributed in a north-south direction, over a length of 100 miles from the crossing of the International Boundary by Flathead river, in the south, to the head of Elk river in the north. "There is, in the southern Rocky mountains of British Columbia and Alberta, a conformable series of sedimentary rocks ranging in age from pre-Cambrian to Upper Cretaceous. These rocks are folded and faulted and at no place is the whole succession exposed. On the Alberta slope of the mountains, the series extends to higher formations than in British Columbia, where erosion since the Rocky Mountain uplift has removed some of the younger formations".

⁽¹⁾ Rose, B., Geol. Surv. of Canada; Sum. Rept. 1916.

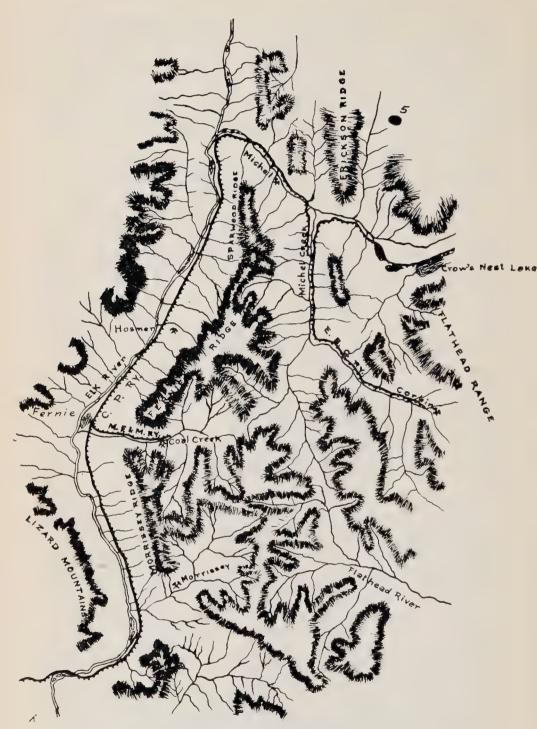


Figure 8.—Crowsnest Pass Coal Field Scale: 6.25 miles to 1 inch.

"Lying conformably on..... [the Palæozoic] rocks, and passing gradually into the Kootenay formation above, is a great shale formation [the Fernie], containing marine fossils of Jurassic age..... At the top the shale becomes more arenaceous and alternating bands of brownish sandstone and shale lead up to a massive, coarse-grained sandstone, above which there are plant remains and coal seams. The base of this heavy sandstone is used as the dividing line between the Fernie shales and the overlying Kootenay formation. The thickness of the [Fernie] formation is over 3,000 feet in the Elk River valley. The formation thins towards the east..... It is commonly found in the valleys about the borders of the coal basins".

"The Kootenay formation contains all the workable coal of the district. It consists of alternating sandstones, shales, and coal seams, with considerable conglomerate towards the top of the section..... The shales are soft and friable, and so they and the coal seams are usually covered with debris.... The succession varies from place to place.... It is... difficult to designate an upper limit to the formation. The character of the strata above is much the same, except that the sandstones are coarser and there are a number of conglomerate bands.... A section measured on the Elk River escarpment north of Morrissey shows 216 feet of coal in 3,200 feet of strata. The lower 2,200 feet of this section contains 205 feet of coal in nineteen seams ranging in thickness from 1 to 46 feet, and above this there is a conglomerate layer which may represent the upper limit of the Kootenay formation. To the north, in the Upper Elk Valley coal basin, the Kootenay reaches a thickness of 3,500 feet, but it thins out to the south and east..... The formation is of Lower Cretaceous age".

"The Kootenay formation grades upwards into conglomerates and coarse sandstones, with shales and thin seams of a semi-cannel nature. These beds have been called the Elk conglomerates...... The strata above the Elk conglomerates are mostly green and reddish shales, and sandy shales interbedded with sandstones and some conglomerates. This part of the series has been called the Flathead beds...... The combined thickness of the Elk conglomerates and the Flathead beds is estimated at 6,500 feet".

"Tertiary lake deposits are exposed along the Flathead river... for.... from 10 to 12 miles northward from the American boundary...... The beds have..... dips of from 15 to 50 degrees".

"The strata of the Rocky mountains lie in a system of parallel folds and faults with a general north-south alignment. This is expressed topographically by a series of parallel mountain ranges with steep faces to the east and gentle slopes to the west. The more resistant rocks.... [the Palæozoic and pre-Cambrian] form the higher ranges, whereas the softer Cretaceous sandstones and shales have been eroded into lower ridges".

The various coal-bearing areas are briefly described in the following, in their order of occurrence from south to north.

Flathead River Coal Areas.—For 24 miles north of the International Boundary, the Flathead river flows nearly due south and at three places are erosion remnants of Kootenay strata. The most southerly commences about 4 miles north of the boundary and extends north for 5 miles. The Kootenay measures, with underlying and overlying Mesozoic strata, form a down-faulted block or graben. The total thickness of the Kootenay along one stream is 1,147 feet, and five coal seams, ranging in thickness from 4 feet to 36 feet, have been found. A second area of Kootenay strata occurs about 10 miles north, and a third 5 miles further north. At the second locality the Kootenay beds lie along the east edge of a downfaulted block. At one place six seams of coal have been found, ranging in thickness from 4 to 40 feet.

Fernie Basin.—"The Fernie basin is the largest connected area of coal-bearing rocks..... Its major structure is that of a broad, flat syncline upturned at the ends. It occupies the mountains east of Elk river and extends 35 miles in a north and south direction..... In width it varies from 4 to 13 miles. The centre of the basin is occupied by the Elk conglomerates and Flathead beds... The Kootenay.... rocks outcrop continuously around the border on the north, west and south sides, but are broken by faulting and folding on the east side.... At Morrissey, twenty-three seams give 216 feet of coal in 3,676 feet of measures. At Fernie twenty-three seams give 172 feet of coal in 2,250 feet of measures. At Sparwood twenty-three

seams give 173 feet of coal in 2,050 feet of lower measures". An estimate of the quantity of workable coal in the basin, made by McEvoy, is calculated on the basis of 100 feet in thickness over an area of 230 square miles and gives 22,595,200,000 tons of 2,240 pounds each. The coal is a high-grade bituminous variety, and does not vary greatly from seam to seam.

Corbin and Other Areas.—"In the area lying between the Fernie basin and the main range of the Rocky mountains to the east, a number of mountains are capped by rocks of the Kootenay formation. In each case a synclinal basin forms the mass of the mountain, but there is much local faulting and folding.... [At Corbin] the coal measures have been very closely folded and faulted and the coal and shale squeezed into pockets on the limbs of the folds. Two mines have been driven on the limbs of a monoclinal fold and in each case the coal seam is over 100 feet thick. Another pocket of coal lies close to the surface.... Diamond drilling shows that this pocket underlies an area of 20 acres with a thickness of from 150 to 200 feet".

Elk River Field.—The Kootenay measures occupy an area which etxends from the head of Elk river southward along the river for more than 25 miles and is continued by an extension of its eastern edge for a further distance of 35 miles. The area of the Kootenay strata of the southward extension of the eastern part of the field, is bounded on the west by a minor range of the mountains and on the east by the main range, both formed of Palæozoic strata. The rocks of the western bounding range have "been thrust eastward and in places faulted over the Jurassic and Cretaceous rocks lying between the two... [mountain] ranges. These [the Mesozoic beds] were folded into a syncline and have been eroded to low hills. Fernie shales flank the two ranges and Kootenay coalbearing rocks occupy the middle of the syncline... The area occupied by the Kootenay is from 2 miles to 1 mile and less in width" [in the southern part]. To the north it extends across Fording river and joins the area bordering the upper part of Elk River valley.

The upper Elk River valley is occupied by the Kootenay beds. The mountain ranges bounding the valley consist of Palæozoic strata. The beds of the western range have been thrust eastward over the Fernie and Kootenay which, over a considerable area, lie in a synclinal form. In places minor faults and folds break the regular arrangement of the rocks. The Kootenay strata occupy a band-like strip of country in places 8 miles wide. The formation is at least 3,500 feet thick and holds a number of coal seams, many of which have a thickness of 10 to 15 feet and some 20 feet. All the coal is bituminous or semi-bituminous. "An estimate made from statements of the thickness of seams supplied by private companies places the probable coal reserve at 12,941,000,000 metric tons" (of 2204.6 lb.).

The coal of the Crowsnest Pass field is a high-grade bituminous fuel, in places becoming anthracite. The majority of the seams furnish a very good coking coal as well as providing an excellent steam fuel. The general nature of the coal is indicated by the following analyses (1):

Mine	Moisture per cent	Volatile matter per cent	Fixed carbon per cent	Ash per cent	Sulphur per cent	B.t.u. per lb.
Michel No. 3	1.4	24.8	62.7	12.5	0.5	13,270
Michel No. 7	1.9	22.6	65.5	11.9	0.4	13,360
Michel No. 8	3.0	24.1	65.7	10.2	0.6	13,480
Coal Creek No. 2	2.2	26.3	64.7	9.0	0.5	13,820
Coal Creek No. 5	1.6	24.0	65.2	10.0	0.5	13,480

Mining Operations in the Eastern District

Only two companies are operating in the district at the present time, the Crow's Nest Pass Coal Company, and Corbin Coal's, Limited, successors to the Corbin Coal and

^{(1) &}quot;An investigation of the Coals of Canada", by J. B. Porter, R. J. Durley, and others; Mines Branch, Dept. of Mines, Ottawa.

Coke Company. The first-named company has collieries at Coal Creek, five miles east of Fernie, and at Michel, on Michel creek about 17 miles directly north of Fernie and about eight miles west of the summit of the Rocky mountains. Corbin colliery is situated on one of the tributaries of Michel creek almost on the interprovincial boundary between British Columbia and Alberta, and about 24 miles east of Fernie.

Coal Creek Colliery.—This colliery marks the site of some of the earliest prospecting for coal in eastern British Columbia. Dr. Selwyn, of the Geological Survey, examined the field in 1891, and in his report he mentions his visit to Coal Creek, where a seam of coal seven feet thick was being worked. This point was later selected as the site for a colliery by the Crow's Nest Pass Coal Company, and it has been in continuous operation ever since, having produced over ten million tons of coal, or about 54 per cent of the entire output of the field.

At Coal Creek, the stream, in its westerly flow, cuts the coal measures, which dip to the east, and several seams are exposed on either side of the creek. The following partial log

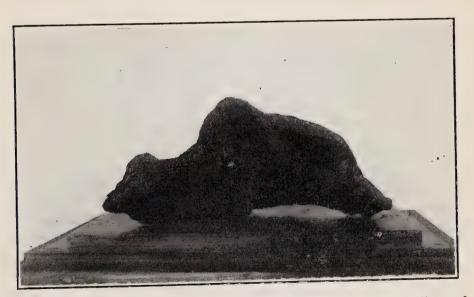


Figure 9.—Piece of coal from Coal Creek, Fernie, showing effect of pressure and thrust.

of a drill hole put down on the south side of Coal creek at the colliery will give an idea of the seams encountered and their position in the measures:

Depth of hole in feet	Coal seam	Thickness of coa
20	'B' seam	5 feet
160	Coal	3 "
219	No. 1 seam	8 to 30 "
349	No. 2 seam	6.5
363	No. 2 lower seam	3 "
416	No. 4 seam (new)	10.5 "
577	No. 5 seam	6 "
640	Coal	2 "
680	Coal	3 "
936	No. 8 seam	9.5 "
1,057	No. 4 seam	17 "
1,167	Coal	5 "
1,178	Coal	3 "
1,184	No. 10 seam	16 "
1,234	Coal	
		4 "
	Total coal	112 feet

At the present time there are five mines in operation, of which three are working on No. 2 seam, and two on No. 1 seam. The mines are driven in from the banks of the creek, on the coal seam, level course, and places set off to the rise and the dip, forming a pillar-and-stall method.

As the mine progresses into the mountain, the thickness of overhead strata gradually increases, until in some places it now amounts to 2,500 feet. Owing to the enormous weight of this overlying material, serious problems have arisen and have had an important influence on the methods of working the mines. With the pillar-and-stall method of working, the pressure from above has caused 'bumps', which besides endangering the lives of the workmen have caused great destruction of mine workings. However, these thick seams do not lend themselves to the longwall system of mining,



Figure 11.—Coal Creek Colliery.

owing to lack of material to fill up the vacant portions of the mine. The result is that it has been found necessary to continue the pillar-and-stall method of working, but the size of pillars has gradually been increased, until not more than fifteen per cent of the coal is being extracted in the first operation. The coal is all mined by hand, and no explosives are used to break it down.

Loading is also by hand, into cars with a capacity of 1.75 tons which are hauled to and from the working faces by horse. Other haulage inside the mine is by either compressedair hoists, endless-rope haulage driven from the outside of the mine, gravity planes, or compressed-air locomotives. last named, which is a very common form of haulage in this district, is practically a large high-pressure tank, to which an engine is attached, running on a frame similar to a mine car but stronger. A high-pressure compressed-air machine situated in the power house delivers air into the mine through pipes at about 1,000 pounds per square inch, and charging stations placed at convenient intervals provide the means of charging the air-tank attached to the locomotive. The locomotive receives the air at a pressure of about 800 pounds, and a reducing valve situated between the tank and the driving engine brings the pressure to about 150 pounds for working purposes.

In some cases, electric motors situated outside the mine operate the haulage underground, but no electric motors of any kind are in use in the mine itself.

Ventilation is provided by large capacity fans, either of the Sirocco or Wilson (Guibal) type, driven by either steam or electric power. These fans deliver an amount of air ranging from \$0,000 cubic feet per minute in the smaller mines to 178,000 cubic feet per minute in the larger. The water gauge varies from one-inch to four-inch, according to the size and the amount of work done by the fan.

That great quantities of air are required in some of these mines will be readily appreciated when it is stated that in one of them 3,640 cubic feet of methane are given out for every ton of coal produced, this figure being based on determinations covering a period of eight years, from 1918 to 1925 inclusive. Another feature that makes efficient ventilation imperative is the occurrence, in this same mine, of 'blow-outs' of methane, in some cases accompanied by the discharge of much fine coal dust. Ten of these blow-outs occurred during the year 1925, and while it is not possible to give more than an approximate estimate of the amount of methane given off on such occasions, judging from analyses of the air in the area flooded, it runs well over one hundred thousand cubic feet, with, in some cases, as much as one hundred tons or even more of fine coal dust.

No. 1 East mine and No. 1 South mine are both working on No. 1 seam, the former on the east, and the latter on the west, portion. Both mines are on the south side of the valley. Mines Nos. 2, 3, and 9 are mining No. 2 seam, which is about 150 feet below seam No. 1. Mines Nos. 2 and 3 are operating on the west, and on the east, portion of the seam, respectively, and both are on the south side of the valley. No. 9 mine is on the north side of the valley.

Figure 12 shows two sections of each of the seams being worked at Coal Creek. Analyses of the coal are also given, and in the case of the extremely thick seam these are of samples taken in sections across the seam. It will be noted that the percentage of ash in analysis No. 4 is very high, and this is

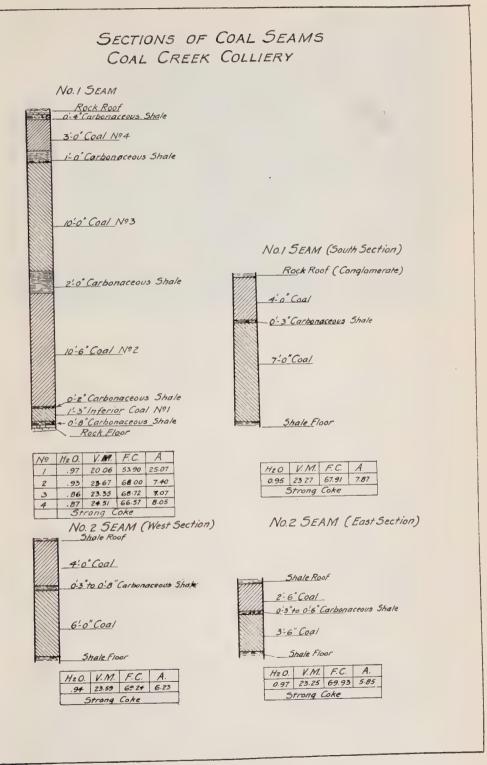


Figure 12.—Sections of Coal Seams, with Analyses. Coal Creek.

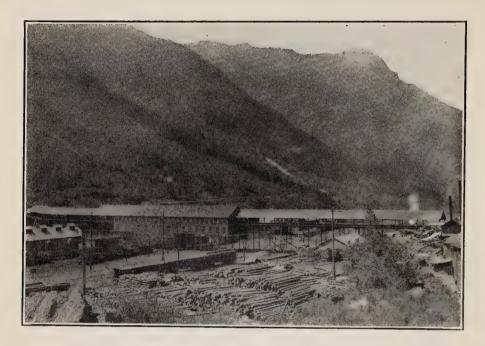


Figure 13.—Coal Creek Tipple.

to be explained by the fact that the sample is from the bottom portion of the seam, which is admittedly inferior coal and not likely to be mined.

The coal from all the mines is brought to a common tipple, which is a steel structure 840 feet in length spanning the valley, and the coal is received from both sides. It is fitted with two independent dumping, screening, and picking installations, all driven by electric motors.

The loaded cars enter a circular dump, positive driven by an electric motor, and this unloads the coal to a feed belt, which supplies it at a regulated speed to the sizing screens. The lump coal passes over the picking table and then is loaded into railway cars for shipment. The fine coal is also loaded into cars and taken either to the coke ovens at Fernie, or sold as slack.

Electrically controlled loading booms allow of the loading of the lump coal into open cars with a minimum of breakage, but as a large quantity of the coal is shipped in box or closed cars, two hydraulic box-loaders are used. These are situated on separate tracks, and consist of a large loading platform, in the form of a cradle, on which the box car is run, then blocked up. The whole platform including the car is then tilted at an angle of 45 degrees, and, by means of a loading chute, the coal is run into one end of the car until that is loaded. The tilting is then reversed to load the opposite end, and so on. This method entails comparatively low labour costs in loading this type of car, which is not very serviceable for the transportation of coal.

Arrangements are made whereby run-of-mine coal can be loaded into open or box cars as required.

The power plant consists of the usual equipment of return tubular boilers, fourteen of these being situated on the north side and five on the south side of the valley, with a joint capacity of about 3,250 horse power.

One high-pressure and three low-pressure compressors are used, the latter to supply power for the pumps and hoists under and above ground, the former to provide for the compressed air locomotives.

Three direct-current generators with a combined capacity of 680 amperes at 250 volts, driven by two Robb-Armstrong steam engines, provide power for the motors on the tipple, and lighting for the colliery and the houses in Coal Creek.

Within the past two years, additional power has been secured from the East Kootenay Power Company, which has hydro-electric plants at Bull river and Elko. This power is being used in driving some of the fans. Future arrangements will call for a greater use of this purchased power to modernize the present mechanical installation at Coal Creek.

As in the collieries on Vancouver island, electric cap lamps of the Edison type are used by the workmen and Wolf safety lamps by the officials, the latter also carrying the Burrel gas detector as an additional safeguard in detecting small percentages of fire-damp.

The colliery is well supplied with the usual buildings incidental to a large colliery, such as lamp room, machine shop, carpenter shop, offices, and warehouses. Large stables are maintained for the horses. More than one hundred of these are used underground, and they are all brought out of the mine at the end of each day's work.

There is a fair sized village, with about 160 Company houses, at Coal Creek, where some of the employees reside. The majority, however, live in Fernie, between which city and the mines the Company maintains a free passenger service on the Morrissey, Fernie and Michel railway, which it owns and operates.

Michel Colliery.—Here Michel creek has cut its channel across the coal measures, and several seams are exposed on each side of the creek. At the colliery, the existence of eleven coal seams has been proved, these being as follows in descending order:

'B' Seam	7 feet
'A' Seam) "
No. 1 Seam) "
No. 2 Seam	3 44
No. 3 Upper Seam8-10) "
No. 3 Seam	
No. 4 Seam) "
No. 5 Seam 8	44
No. 7 Seam 8	66
No. 8 Seam 12	66
No. 9 Seam. 14	4.6

THE CROWS NEST PASS COAL CO LTD.

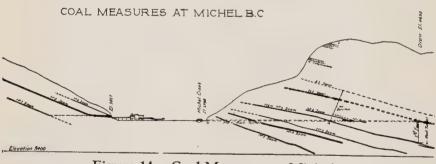


Figure 14.—Coal Measures at Michel.

A cross-cut tunnel driven on the west side of the valley cuts seams Nos. 5, 4, 3, Upper 3, 2, and 1, and it is being continued at the present time to reach seams 'A' and 'B', which have only recently been exposed on the surface.

On the east side of the valley, Nos. 7, 8, and 9 seams have been worked at various times, but at present only No. 8 seam is being mined here. The Upper No. 3 seam is the only one being mined on the west side.

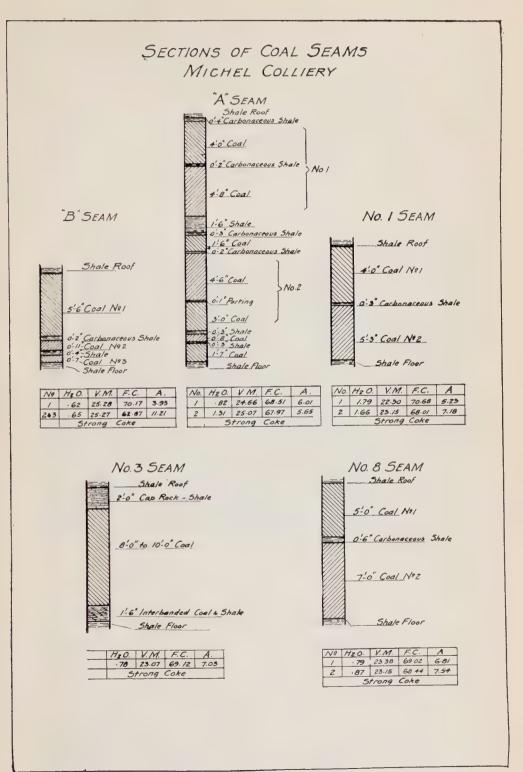


Figure 15.—Sections of Coal Seams, with Analyses. Michel.

At No. 3 mine, the method of work is pillar-and-stall, pillars being left about 50 feet square and stalls being driven 12 feet wide. The coal is mined by hand and is either broken down or blasted and then loaded into mine cars of 1.5-tons capacity. The seam has a fairly uniform dip of about 15 degrees to the west, and the rooms are driven across the pitch.

Horses collect the loaded cars and haul them to the incline or slope as the case may be, from which point they are hauled up or lowered down to the levels, and from there taken by compressed-air locomotive to the tipple. Ventilation is provided by fans of large capacity, similar to those in the other mines described, and here also it is a necessity that large quantities of air should be circulated around the working faces, as the amount of methane given off is fairly large. For the No. 3 seam it averages well over 5,000 cubic feet per ton of coal mined, while in the Upper No. 3 seam, the one at present being worked, the average is about 1,400 cubic feet per ton of coal mined.

The other mine in operation at Michel Colliery is No. 8. situated on the east side of the valley, where the coal seam is easily traced, rising with the mountain side at an angle of between 15 and 20 degrees. This mine is working the southern extension of the seam. The northern portion, which was mined for many years, and convenient to which the tipple is situated, had to be sealed off due to a fire inside the mine. The entrace to No. 8 mine is about 500 feet above the valley where the tipple is situated, and is reached by a self-acting or gravity incline, about 1,000 feet long. No. 8 seam varies in thickness from 5 feet to, in some places, over 30 feet, each thinning of the seam being followed invariably by an increased thickness. The coal is of a dark grey colour, and very similar in appearance to that of No. 1 seam at Coal Creek, breaking along pressure slips and inclined to be friable. being a good steam coal, it is very much in demand for its coking qualities.

The methods of mining, timbering, haulage, etc., are similar to those in the other mines described.

The coal from this mine is taken to a small tipple situated at the same level as the mine, where it is weighed and dumped into a storage bin. From this bin, it is loaded into an eight-ton automatic dumping skip, which conveys it to another storage bin on the same level as the main tipple, to which it is taken, after being drawn into ordinary mine cars, by an endless haulage The main tipple is of the Green patent double-track type, one track being built exactly above the other.

The loaded cars from the west side (No. 3 mine) are brought to the tipple by air locomotives, and a chain-haul feeds them to a double chain-haul fitted with cross-bars, which grip the wheels of the loaded cars and push them up the incline.



Figure 16.-Michel Colliery, from the east.

The dump for unloading the cars is simply a continuation of this track at an angle of from 60 to 70 degrees, and as the car ascends, an end door is opened, which allows the coal to run out of the car into a chute. The empty car continues to travel with the chain haul, and is pushed into a swing transfer lift, while the chain haul travels around a large sprocket wheel, carrying the rear end of the car with it, to the overhead track, which it descends, the cross-bars of the chain acting as a retarder.

This arrangement for hauling the cars to the tipple and unloading them is now being replaced by a more simple method, which consists in unloading the cars at the foot of the incline and taking the coal directly to the sizing screens by means of a belt conveyor, during which process the coal can be picked to some extent. A similar system, with the operation reversed, is in use on the east side to handle the coal from No. 8 mine, the loaded car entering the tipple on the upper track and returning on the lower.

The tipple is equipped with screens and picking tables for sizing and picking the coal, loading booms, open and box car loading arrangements, etc., similar to those at Coal Creek colliery. The slack coal is taken by a belt conveyor direct from the sizing screens to the storage bins used in connection with the coke ovens.

In contrast to Coal Creek, where the coke ovens are situated five miles from the colliery, at Michel they are built convenient to the mine. There are 440 ovens of the bee-hive type, but owing to the present demand for coke being relatively small, only about one-half of these are working. The slack is drawn from the storage bins into lorries, six or eight tons as required, and hauled to the coke ovens by steam locomotive. A narrow-gauge track runs between double rows of the ovens, so that ovens on either side can be charged direct from the lorries. The time required for burning varies from 36 to 56 hours depending on the size of the charge, which is either six tons or eight tons per oven. No attempt has been made to recover any of the by-products.

The power plant consists of eleven return tubular boilers, fired by hand, with a combined capacity of 1,600 horse power. Two low-pressure, and one high-pressure, compressors provide power for pumps and hoists underground, and the high-pressure machine also supplies power for the compressed-air locomotives. There are also two steam-driven generators, which formerly provided power for the tipple, and light for the plant and the town of Michel; but the greater part of the power for these purposes is now obtained from the East Kootenay Power Company.

Michel colliery is well equipped in every way for carrying on the business of producing coal and coke. The town of Michel, owned by the Crow's Nest Company, has grown with the development of the colliery, and now boasts over 220 houses. The majority of the workmen reside in Michel and the remainder in Natal, a small town situated one mile to the west.

The Morrissey, Fernie and Michel Railway Company, a subsidiary of the Crow's Nest Pass Coal Company, owns and operates the track which connects the colliery with the line of the Canadian Pacific railway. It formerly connected also with the Great Northern railway, but the tracks of this railway. both to Fernie and Michel, have been abandoned, and the Great Northern has running rights over the Canadian Pacific railway, with connections to their own branch at Elko, twenty miles west of Fernie.

Figure 15 shows sections of seams 'A', 'B', No. 1, No. 3, and No. 8, and also gives the results of analyses of the coals.

Corbin Colliery.—This colliery, opened in 1908, is owned and operated by Corbin Coals, Limited, successors to the Corbin Coal and Coke Company. It is situated about twenty miles east of Fernie, and near the Alberta-British Columbia boundary. The Company controls an area of about 20,000 acres, and has opened several mines on the property.



Figure 17.—Corbin Colliery, showing town and tipple, also storage piles.

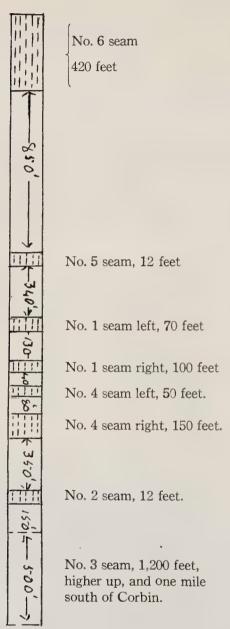


Figure 19.—Horizontal section through Coal Mountain, Corbin.

Corbin is reached by the Eastern British Columbia railway. which connects with the Canadian Pacific railway at McGillivray, fourteen miles distant. The town is 5,000 feet above sea level

As stated above in the section on the geology of this district, the coal measures in the Corbin area have been very closely folded, and the coal and shale squeezed into pockets on the limbs of the folds, where diamond drilling has shown thicknesses of coal up to 200 feet. In one horizontal section of 2.604 feet, seven seams, with an estimated aggregate thickness of 814 feet of coal, have been noted.

The first mine opened was on the No. 1 seam, which at first showed a thickness of 170 feet; but, as development proceeded, a band of shale intervened, splitting the seam in two parts respectively 100 and 70 feet in thickness.

Later, operations were commenced on one of the large 'pockets' of coal, situated about one mile south of Corbin and about 1,000 feet higher in elevation than the town. In working this large deposit, the over-burden, which mainly consisted of coal blossom, was stripped by steam shovel and loaded into railway dump cars, which were hauled away and emptied clear of the coal. Another steam shovel loaded the coal directly into railway cars, the shovel working a bench of coal varying from fifty to seventy feet in height. The Eastern British

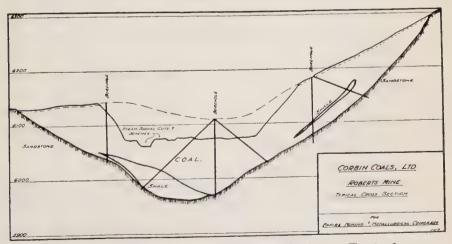


Figure 20.—The Roberts, or No. 3, Mine, Corbin. Typical crosssection.

Scale: 1 in. = about 210 feet.

Columbia railway was extended to reach this deposit, locally known as the Roberts or No. 3 mine, climbing the hill very rapidly by means of several switch-backs, the usual engine being replaced by a Shay locomotive.

This large deposit of coal is in direct line with the No. 1 seam at Corbin, and although the geological relations of the two have not been fully established, drill holes and open-cuts would indicate that it is a continuation either of the No. 1 seam or of some other seams existing at Corbin.

The mine opened on No. 1 seam was worked on a pillar-and-stall method adapted to a vertical vein, four levels being driven on a horizontal plane and connected by cross-cuts. Raises were set off every fifty feet, and when these reached fifty feet, they also were connected by cross-cuts, thus practically cutting the seam up into fifty-foot cubes. This work was continued until the outcrop was reached, which was practically on the surface, when extraction of the pillars commenced on the retreating system. Later this was changed to a caving method, the pillars being split up, and allowed to run, either by the breaking up of the coal itself, or by the crushing of the sides. Small manholes have occasionally to be driven, and the action of caving assisted by blasting. The coal is taken out through chutes on the main level, into cars with a capacity of 1.5 tons, and hauled to the tipple by horses.

A few years after it was opened, this mine went on fire, the cause being either spontaneous combustion or an ignition from a bush fire outside the mine, and the mine had to be abandoned.

Four other seams close to the Corbin camp have been mined to some extent. These are known as No. 5 and No. 2, both about twelve feet in thickness, and No. 4 and No. 6, the first about 200 feet and the other over 400 feet, thick. Of these, the greatest development has been on No. 4, which was worked by the same method as No. 1 seam.

Close examination of No. 6 seam shows a total thickness of 436 feet, but with many bands of shale, which reduce the actual coal to 278 feet. At present this mine is not operating. The seam will probably have to be mined open-cast with steam-shovel, extracting the whole seam between the two walls, and then eliminating the foreign material by a separation



Figure 22.—The Roberts, or No. 3, Mine, Corbin. Coal face and steamshovel bench.

method of cleaning. Careful sampling and analysis have shown that the average ash content of the whole seam is about 17 per cent, and it is believed that it will be cheaper to mine the whole seam on a large scale, and depend on the separation plant to eliminate the foreign material, than to attempt selective mining. As regards mining, timbering, etc., the mine is similar to those already described. Ventilation has not proved a very difficult problem, owing to the seam outcropping at so many points.

Where such great thicknesses of coal exist, it is usual to find a greater amount of associated foreign material than where the seams are thinner, more regular, and well stratified. The problem at Corbin, therefore, is not so much concerned with the mining of the coal as with cleaning it, to make it

suitable for the market.

Up to the present, apart from the usual separation of lump coal from the finer sizes, by hand picking, the preparation of the coal has not received much attention in this district. At Hosmer mines, a washer was installed to wash the slack for the coking plant, but at Corbin the washed coal is shipped to the market, in some cases hundreds of miles away, and the shipping of wet coal during the severe winter months is a problem which has not so far been entirely solved.

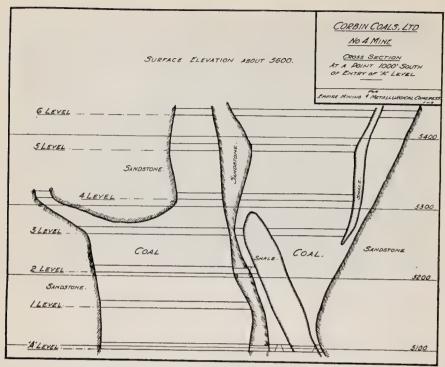


Figure 23.—No. 4 Mine, Corbin. Cross-section. Scale: 1 in. = about 165 feet.

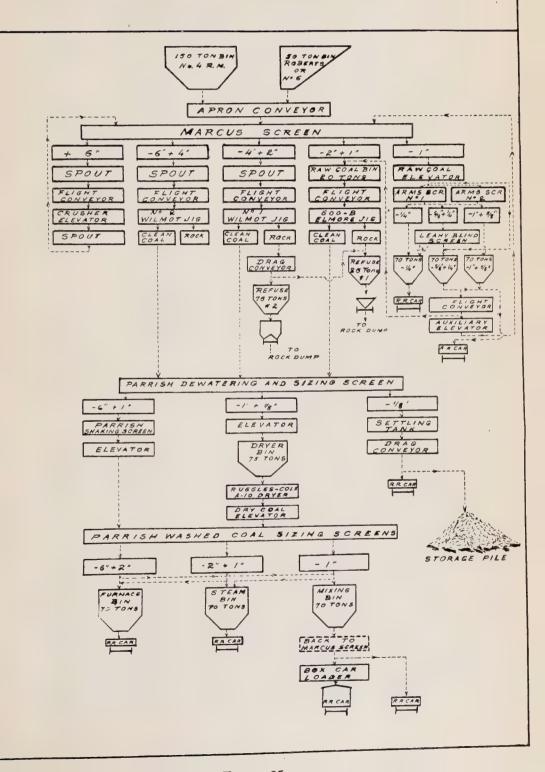
The coal at Corbin is in many respects different from the coal mined in the other parts of the Crowsnest field. It has not proved suitable for coking, it runs low in volatile matter and high in fixed carbon and ash, and it is shiny black in colour. An average analysis gives: moisture, 0.75 per cent; volatile combustible matter, 20 per cent; fixed carbon, 69 per cent; and ash, 10.25 per cent.

A plant for washing and separating the coal from the admixed foreign matter has been erected, and although not quite complete is already treating the present output of the mines successfully. The practice of large-scale mining and preparation of the coal has only been adopted after a great deal of investigation and research, and the working of the scheme will be watched with interest.

The coal from the underground mining is dumped into a chute which feeds an apron conveyor delivering the coal to a Marcus screen. The coal from the Roberts or No. 3 mine is also delivered into the same hopper chute.

CORBIN COALS Lted.

SIMPLIFIED FLOW SHEET
SHOWING THE PREPARATION OF CORBIN COAL
WET WASHING PROCESS



The Marcus screen makes the following sizes: minus 6-inch and over 4-inch; minus 4-inch and over 2-inch; and minus 2-inch and over 1-inch. The large size goes to a No. 2 Wilmott, or simplex-type, jig; the second size to a No. 1 Wilmott jig; and the third size to an Elmore jig. The fine coal, below one-inch, is separated by two arm screens, and the product between 1/4-inch and 1-inch is passed to the Elmore jig. The smaller size, below 1/4-inch, is stored in bins for re-mixing.

The washed coal from the jigs is passed to a Parish de-watering screen, sizes over one-inch being by-passed to shaking chutes to the elevators and sizes below one-inch being passed through a Ruggles-Coles dryer, where the material is dried to about 3 per cent moisture. All washed coal is then elevated to a sizing screen, which again sizes it previous to entering the bins, or delivers it into a central mixing bin for making mine-run product.

The plant at present in course of erection will be able to handle about 1,000 tons per day.

The coals to be treated vary considerably in grade. Thus three samples contained respectively: 18.4, 23.9, and 25.1 per cent of ash. After washing, these samples gave products containing 11.9, 14.0, and 12.0 per cent of ash. These tests were made with the Elmore jig, and it will be noted that the reduction in the ash content ranges from 35 to 52 per cent, and leaves a uniform mechanically cleaned product with an ash content not exceeding 14 per cent.

The coal-cleaning problem at Corbin is a comparatively simple one, as there is little shale or bone mixed with the coal, such as would necessitate crushing and separation by tables. The extraneous matter is quite distinct from the coal, making the problem one of straight separation.

Future developments at Corbin will probably be along the lines of mining these huge seams and deposits of coal on a large scale by some method of stripping and loading with steam shovels, and depending on the cleaning and preparation plant to deal with the output and place it on the market in a suitable condition. Trouble has been, and will be, encountered during the winter owing to the heavy snow fall, but the storage, convenient to the washing and screening plant, of large quantities of coal mined during the summer will probably help

to tide over the winter period. To this will be added a certain amount of coal mined underground, for underground mining will probably continue to contribute a portion, even if a small one, of the coal production.

The steam plant at Corbin consists of one Babcock and Wilcox and three return tubular boilers, which provide the steam to drive the electric generators.

All the machinery is driven by electric motors, except an outside hoist driven by steam.

The colliery is equipped with the usual machine and carpenter shops, warehouses, and offices incidental to the operation of a fairly large colliery. A wash or change room is provided for the employees, all of whom reside in the town of Corbin, which is owned by the Company.

The following table shows the production of coal from each colliery operating in the Eastern District during the year 1925, the number of employees, and the amount of coal produced per employee per day:

Colliery	Production (long tons)	Number of employees	Production per employee per day
Coal Creek	464,133	760	2.23
	321,535	582	2.37
	68,812	124	2.84

The other two collieries formerly operating in this district, the Hosmer and Morrissey—the former owned by the Canadian Pacific Railway interests and the latter by the Crow's Nest Pass Coal Company—have been dismantled. This was due to economic considerations, and not to any lack of coal. Doubtless in the future, mining engineers will solve the problem of mining these coal lands both profitably and economically.

THE CENTRAL DISTRICT

Historical and General

The Central District of British Columbia as defined for the purposes of this article, includes: (1) the coal fields of the region lying between the Rocky mountains on the east and the Coast range on the west; and (2) the coal field in the Peace River country in northeast British Columbia, east of the Rocky mountains. The Central District as thus defined, includes the whole length of the province, nearly 1,000 miles in a northwest-southeast direction, and a breadth of country exceeding 250 miles. Within this great stretch are many coal fields of varying relative importance.

At the present time, practically the whole of the coal production of this district is coming from the Nicola and Similkameen mining divisions. During the past two years there has been a small output from the Bulkley River (Telkwa) area, and some years ago a few thousand tons were mined in the North Thompson or Chu Chua area.

Dawson mentions the existence of coal both in the Nicola and Similkameen valleys in his report on the preliminary survey he made of the district in 1877-78, but it was not until 1906 that mining on a commercial scale commenced. Doubtless coal was being mined prior to this for use locally by farmers and other settlers in the districts, but lack of railway facilities prevented development on a large scale. The building, in 1906, of a branch line from Spence's Bridge, on the main line of the Canadian Pacific railway, to Nicola, gave direct access to the Nicola valley, and two years later the Nicola Valley Coal and Coke Company commenced active mining of coal, producing about 25,000 tons in that year.

In the Similkameen valley, active production commenced in 1910, with the arrival of the Victoria, Vancouver and Eastern railway, now a section of the Kettle Valley railway.

The principal mines in the Central District have been in the vicinity of Merritt (Nicola area) and Princeton, but during the past few years Coalmont colliery, situated twelve miles west of Princeton, has become the principal producer of coal in the Similkameen valley. The following table shows the total amount and value of the coal produced to date in this district:

Coal Area	Production (long tons)	Value · \$
NicolaSimilkameenChu ChuaBulkley River (Telkwa)	2,108,057 996,742 3,085 5,157 123	7,905,213 3,737,782 11,568 19,338 461
Total	3,113,164	\$11,674,362

The nature of the coal varies in different parts of the district, being lignite in the Similkameen valley, semi-bituminous in the Nicola area, bituminous in the Telkwa area, and anthracite in the Groundhog field in the north.

While the limited amount of work, both geological and actual mining, makes it difficult to estimate with any degree of accuracy the amount of coal available in the productive portion of the Central District, the following may be taken as a rough approximation:

NicolaPrinceton	400,000,000 long tons 435,200,000 " 110,636,000 "
Total	

Geology of the Central District (1)

The several coal fields of the Central District, with the exception of that of Peace river, lie within a single geological province whose foundations consist of Jurassic and older sedimentary and volcanic strata, folded, faulted and invaded by batholiths of granitic rocks of late Jurassic and early

⁽¹⁾ Prepared by Dr. G. A. Young.

Cretaceous age. In western districts, coal-bearing Cretaceous strata lie on the older rocks. These also have been subject to mountain-building forces. Prior to an early period of Tertiary time, the region in general was transformed by erosion to a comparatively level condition. In late Eocene or early Oligocene time a period of sedimentation and volcanic activity commenced and which has continued intermittently to recent time. The earlier Tertiary sediments of various districts are coal-bearing. Locally the Tertiary beds have been considerably deformed by folding and faulting but on the whole they are flat-lying. Volcanic rocks predominate but the once buried coal-bearing horizons have been locally exposed by erosion resulting from regional uplift.

The Peace River area lies east of the Rocky mountains and belongs to another geological province which embraces the coal fields of Alberta and the already described Eastern District of British Columbia. This coal field and the others belonging to the Central Division are briefly described on following pages in the order of their geographical position, commencing with the most southerly, but omitting various areas in which the coal seams so far discovered are apparently of little or no economic importance.

White Lake Coal Area (1):

White Lake coal area is in Okanagan valley about 6 miles west of Okanagan Falls. The field has an area of about 6 square miles and "is a basin-shaped depression.... No rocks older than Tertiary are exposed in or around White Lake area... The coal-bearing rocks.... consist of tuffaceous sandstones and true tuffs, shales, conglomerate, breccia, and thin seams of coal. A section along the valley of Prather creek..... gave a thickness of about 2,000 feet of beds.... The lowest third of the section contains a preponderance of black and grey shales.... associated in places with thin seams of coal...... In the central part of the area some grey shales and two narrow seams of coal outcrop. These beds..... [probably overlie those of the measured section.]..... So far as at present known, the most important seams of coal are those exposed in a small

⁽¹⁾ Camsell, C., Geol. Surv. of Canada; Sum. Rept. 1916.

ravine on the northwest side of White lake. These seams are respectively 14 and 20 inches in thickness.... [The] coal is of a bituminous character..... In general the structure.... is that of a synclinal basin.... The dips range from 0 degrees to 50 degrees and average about 30 degrees. Some faulting has taken place, especially in the disturbed region on the east".

Princeton Coal Area (1):

Princeton is situated in a shallow depression underlain by coal-bearing Tertiary sedimentary strata. The sediments occupy an area of about 50 square miles. Along the eastern side of the area, the underlying pre-Tertiary rocks outcrop, but elsewhere, along most of the boundary, the sediments dip beneath Tertiary lavas. Tertiary igneous rocks also occur cutting the sediments. The Tertiary sediments consist of sandstone, shale and seams of lignite. The strata lie in a series of folds but dip at low angles. Lignite seams outcrop in various places and have been penetrated by drills. In the vicinity of Princeton one seam has a thickness of more than 18 feet and in 90 feet of strata seven seams occur with an aggregate thickness of 341/2 feet. Elsewhere seams of 9 and 10 feet have been found.

The Princeton coal area is in the Similkameen mining division.

Tulameen Coal Area (2):

Tulameen coal area, also in the Similkameen mining division, lies about one mile south of Tulameen and twelve miles northwest of Princeton. The field is oval of outline and has an area of 5 square miles. The coal-bearing Tertiary sediments rest on pre-Tertiary rocks and, in places, on earlier Tertiary volcanics. About one-third of the field is covered by a sheet of basalt. The beds have a general synclinal structure complicated by minor folds. The average angle of dip is about 40 degrees. The sediments have a thickness of somewhat less than 2,500 feet. The lowest part, 600 feet thick, is composed mainly of sandstone; the middle part, 460 feet thick, is of shale, with which occur the principal coal seams; the

⁽¹⁾ Camsell, C., Geol. Surv. of Canada; Pub. No. 986. (2) Camsell, C., Geol. Surv. of Canada; Mem. 26.

upper part is largely of sandstone. On the south edge of the basin, where the coal is best exposed, there are at least four seams of workable coal, $6\frac{1}{2}$, 3, 5, and $5\frac{1}{2}$ feet thick respectively. These lie in about 120 feet of strata. It has been estimated that the field contains more than 65,000,000 tons of coal that can be extracted. The coal varies in character from one part of the field to the other. It is a bituminous variety or, as in places, of somewhat lower grade.

Nicola and Quilchena Coal Areas(1):

The Nicola and Ouilchena coal areas lie in the basin of Nicola river. The two fields are occupied by early Tertiary shale, sandstone, conglomerate, and coal, that rest on disturbed pre-Tertiary strata and are overlain, in places, by Tertiary basalt flows. The Nicola field lies in the vicinity of Merritt and has an area of about 40 square miles; the Quilchena field is 10 miles east and has an area of about 10 square miles. The strata of the Nicola field in places lie in folds, in other places are faulted, and dip at angles varying from 10 to 40 degrees. The number of the workable seams of coal contained in the basin is not definitely known but, in the vicinity of Coal gully, four seams occur in a thickness of about 400 feet of strata and, where mined, have thicknesses of 6, 10, 5, and 12 feet respectively. The coal is a low-carbon, bituminous variety. In the smaller, Quilchena field, to the east, seams of coal, 4 to 6 feet thick, are reported to outcrop.

Hat Creek Coal Area (2):

The Hat Creek coal field lies in the valley of Hat creek, 15 miles west of Ashcroft. The coal-bearing, early Tertiary strata occupy an area 15 miles long and several miles wide. The beds rest on pre-Tertiary rocks and in places along the slopes of the valley pass beneath Tertiary volcanic rocks. The Tertiary sediments consist largely of shale and sandstone and may be 1,000 or more feet thick but are very poorly exposed. The bordering, overlying Tertiary volcanics have been subjected to folding and faulting, and dip at angles as high as 45 degrees.

⁽¹⁾ Camsell, C., Twelfth Inter. Geol. Cong., Guide Book No. 9; issued by Geol. Surv. of Canada. Ells, R. W., Geol. Surv. of Canada; Sum. Rept. 1904. (2) MacKay, B. R., Geol. Surv. of Canada; Sum. Rept. 1925, Part A.

The strata immediately associated with the coal outcrops, show faulting and minor folding, and dip at angles as high as 70 degrees. Owing to the general lack of exposures, the structure of the field and the thickness and stratigraphic order of the coal seams is unknown. A tunnel has pierced 155 feet of beds of which 108 feet is clean coal, 35 feet is shaly coal, and 12 feet is shale; neither hanging-wall rock nor foot-wall rock was encountered. A bore-hole penetrated 428 feet of strata of which 270 feet was coal and 37 feet shaly coal; the coal occurred in sixteen beds ranging from 1 to 211 feet in thickness separated by shale zones 1 to 26 feet thick.

The coal is a lignite. It has been calculated that within the area penetrated by borings, about 100 acres, there are 16,800,000 tons of clean coal.

North Thompson or Chu Chua Coal Area (1):

The North Thompson or Chu Chua coal field is in the valley of North Thompson river, 50 miles north of Kamloops. The coal-bearing strata are of late Eocene or early Oligocene age and occur in isolated outcrops mainly along the bottom and lower slopes of North Thompson River valley. The strata are known to be coal-bearing at one locality, where they may extend 5 or 6 miles along the valley but are exposed only along the banks of a small stream. Along this stream, the strata dip at angles of about 25 degrees, may be 2,500 feet or more thick, and consist of conglomerate overlain by coarse sandstone, arkose, and sandy shale, with three main coal seams and a number of minor seams. The lowest of the three main seams is nearly 4 feet thick; the next above consists of two benches, 10 inches and 18 to 20 inches thick respectively, separated by 2 feet of sandstone and shale; the highest consists of 8 to 30 feet of sandstone from a bench varying from 30 inches of clean coal to 1 foot or more of coal and sandy shale.

The coal seams thin and thicken, split and unite, in short distances. The coal is a low-grade, low-rank bituminous or low-grade, high-rank sub-bituminous variety.

⁽¹⁾ Uglow, W. L., Geol. Surv. of Canada; Sum. Rept. 1921, Part A.

Bowron River Coal Area (1):

The Bowron River coal area is 45 miles east of Fort George, "The coal measures [of Tertiary age] lie in a flat basin surrounded by hills.... of crystalline rocks..... A very limited section of the measures is exposed [along Bowron river] but at one place three workable seams outcrop". The seams are, respectively, 12 feet thick with 7 feet 8 inches of coal, 5 feet thick with 4 feet 2 inches of coal, and 10½ feet thick with 9 feet 2 inches of coal. "In the measures immediately below... numerous thin seams of coal occur, up to 3 feet in thickness, separated by thin bands of shale and sandstone..... Other thin seams appear elsewhere, apparently at a slightly higher horizon.... The coal is bituminous.... At the point where the three workable seams are exposed, the measures dip at an angle of about 43 degrees but.... the dip moderates towards the northeast, so that the measures may reasonably be expected to lie comparatively flat under the greater part of the area".

Other Areas of Tertiary Coal-Bearing Strata:

Tertiary sediments carrying coal, occur at other localities in the Central District. In most cases the coal seams so far found in such areas are thin or but little is known of their thickness and extent. The following is a list of some of these localities: the head of the North fork of Kettle river, west of Midway, and in the vicinity of Kamloops lake, all in southern British Columbia; Blackwater river, Fraser river near Fort George and in the vicinity of Quesnel, Nechako river, and Kohasganko river a branch of Dean river, all in central British Columbia: Parsnip river, Liard river, and Tuya river in northern British Columbia.

Skeena River Basin Coal Areas:

A number of detached areas of Cretaceous strata in the basin of Skeena river, northwestern British Columbia, are coal-bearing. These detached areas occur over a length of 230 miles, from the Groundhog coal field in the north to the Morice River area in the south. The coal-bearing strata belong to the Skeena series, which consists of conglomerates, sandstones, shales, and coal, and is in places or in part of marine

⁽¹⁾ Galloway, C. F. J., Canadian Min. Jour., Vol. 33, p. 335.

origin. The Skeena beds in most localities are considerably disturbed and rest on Jurassic measures which, in neighbouring districts, are invaded by batholithic, granitic bodies, but these intrusives are not known to cut the coal-bearing Skeena series. The following descriptions are of various coal fields within the Skeena River basin.

Groundhog Coal Field.—This field (¹) lies 140 miles north of Hazelton on the headwaters of Skeena river and of streams flowing to the Nass and Stikine rivers. Its area is about 900 square miles but "over large areas the coal seams have been removed by erosion and.... in other areas, only a very small fraction of the total number of seams has been preserved". The area is mountainous. Jurassic strata "outcrop along anticlines on both sides of the Skeena-Stikine valley and surround the field on all sides.... The geological structure... is complex..... In general the strata appear to lie in folds overturned to the northeast.... [and], in general, dip to the southwest.... The main folding in many places is complicated by minor folds and crumples.... [and] by pronounced faulting".

The Skeena series consists of shales, sandstones, and conglomerates having a total thickness of 3,900 feet or more. Coal, in seams up to perhaps 20 feet thick, occurs at many horizons throughout the total thickness of the series and there are indications that many of the seams are persistent. In one section, measuring 3,944 feet, eighteen coal seams have been noted having a total thickness of clean coal of 34.2 feet, the thickest seam measuring 12 feet. The coal is anthracite.

Sustut River Coal Field.—This area (2) lies southeast of the Groundhog field. The strata belong to the Skeena series and in them several coal seams, 2 to 4 feet thick, of lignitic coal, have been discovered.

⁽¹⁾ Malloch, G. S., Geol. Surv. of Canada; Sum. Rept. 1912.

⁽²⁾ Malloch, G. S., Geol. Surv. of Canada; Sum. Rept. 1912

Kispiox and Shegunia Coal Field.—This field (1) occupies an area of about 10 square miles along Skeena river, 8 miles north of Hazelton. The strata belong to the Skeena series and are much disturbed. In them have been found coal seams, 3 or 4 feet thick or thinner.

Babine Lake Coal Field.—The Skeena series with seams of coal, occurs on Tuchee river flowing into the west side of Babine lake (2).

Zymoetz River Coal Field.—An area of the Skeena series occupying about 25 square miles borders Zymoetz river near its head. The strata for the most part dip at low angles but are flexed and slightly faulted. On Coal creek, five coal seams have been found close together. The thickest seam has 6 feet of clean coal separated by 7 inches of clay from 3 feet of clean coal. The coal is of a good bituminous grade (3).

Bulkley River Coal Fields.—The coal-bearing Skeena series in the Bulklev River areas has a thickness of 600 to 800 feet and consists mainly of shales and sandstones. "These beds occur in a number of comparatively small patches in various widely separated localities.... The most important localities are situated on the Telkwa river and the headwaters of the Morice (4)". In some of the areas the strata are greatly disturbed. The coal of different areas varies from a lignite to a semi-anthracite. The quality of the coal seems to depend on the proximity of the sediments to certain intrusive bodies of igneous rock. On Coal creek, at the headwaters of Morice river, the coal is anthracitic and seams of 4 to 7 feet thickness have been found. Similar coal occurs in a nearby area. Both fields are only a few square miles in extent. A comparatively large area of the Skeena series occurs on Morice river. In it various seams have been found, one of which measures 10 feet. Various small basins of the coal-bearing strata exist on the Telkwa river with coal seams up to 12 feet thick.

 ⁽¹⁾ Leach, W. W., Geol. Surv. of Canada; Sum. Rept. 1909.
 Malloch, G. S., Geol. Surv. of Canada; Sum. Rept. 1911.
 (2) Leach, W. W., Geol. Surv. of Canada; Sum. Rept. 1909.
 (3) Galloway, J. D., British Columbia Dept. of Mines, Bull. 4.
 (4) Leach, W. W., Geol. Surv. of Canada; Sum. Repts. 1907, 1908, 1910.

and 1910.

Peace River Coal Area (1):

The Peace River coal area lies in northeastern British Columbia, east of the Rocky mountains. The coal-bearing strata belong to the Cretaceous area of northern Alberta. In the foot-hills of the Rocky mountains, where crossed by Peace river, they rest on Triassic marine strata. The coal-bearing beds locally dip at angles as high as 45 or 50 degrees, but for the most part dip at angles lower than 10 degrees. The lowest member of the Cretaceous section is the Bullhead Mountain formation, 4,400 feet or more thick. The lower part is composed of conglomerate and sandstone; the upper part of sandstone, shale, and coal. The strata are of Lower Cretaceous age and probably of about the same age as the coal-bearing Kootenay formation.

A few thin coal seams occur in the lower part of the Bullhead Mountain formation. In the upper part, in Peace River canyon, there are 60 or more coal seams in 1,250 feet of strata. Most of the seams are small. Eight are known which have a thickness of 2 feet 6 inches to 4 feet 8 inches, one varies from 5 feet 5 inches to 5 feet 9 inches, and another from 3 feet 7 inches to 8 feet 4 inches. The coals vary in grade from bituminous to semi-bituminous.

Mining Operations in the Central District

Nicola Coal Area:

Middlesboro Colliery.—This colliery, owned by Middlesboro Collieries, Limited, is situated near Merritt, in the Nicola valley. Three mines are at present working.

The mines were opened from the outcrop, but later cross-cut tunnels were driven in, to cut off the outside work and reduce the haulage. The method of mining is pillar-and-stall, the coal being mined by hand and either loaded directly into cars or carried down the rooms by chutes and then loaded. Timbering, haulage, and pumping, follow the methods described for other mines of these districts, compressed air being used for

⁽¹⁾ McLearn, F. H., Geol. Surv. of Canada; Sum. Repts. 1917 Part C; 1920, Part B, and 1922, Part B.

hoists and pumps. Ventilation is provided by small fans of the Sirocco type, some of them made locally, and fire-damp has not proved a very serious factor to deal with.



Figure 26.—Tipple, Middlesboro Collieries.

Considerable trouble has been met with as a result of the extensive faulting, which appears to become more excessive with increasing depth of the seams in the measures, and the quality of the coal has deteriorated to some extent as this faulting increased. This feature of the seams has necessitated a great amount of prospecting work, diamond drilling and the driving of many small tunnels, which has greatly increased the cost of producing the coal.

The loaded coal is brought to the tipple in cars of 1.5 tons capacity and is screened and picked at a central tipple before being shipped.

The power plant consists of four return tubular boilers, a compressor, and two generators, one for power and the other for lighting.

A small camp is maintained near the colliery, but the majority of the workmen reside in the town of Merritt, about one mile distant.

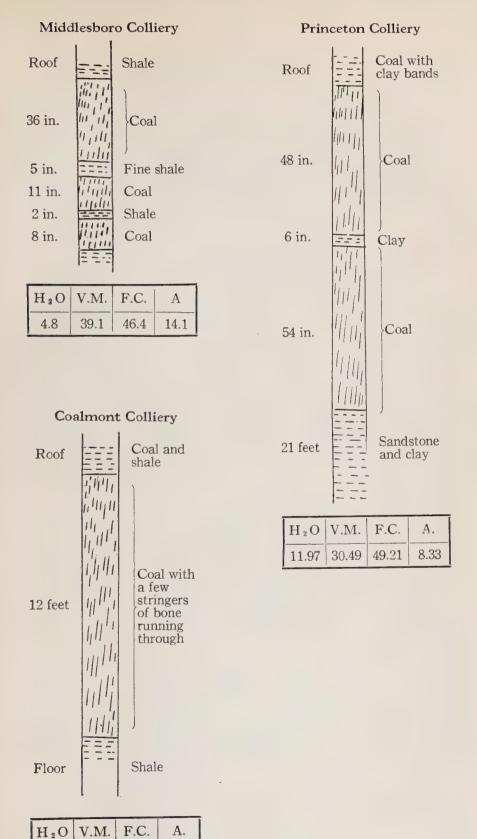


Figure 27.—Sections of Coal Seams, with Analyses. Nicola-Similkameen Coal Field.

2.3

54.0

33.1

12.3

The other collieries in this field are small, with very primitive plants, and the methods of working are very similar to those at the Middlesboro colliery.

The mines in this area are all connected to the Canadian Pacific railway by the Kettle Valley railway. The principal point for disposal of the coal is Vancouver, where it comes in competition with the better prepared coals of Vancouver island.

Similkameen (Tulameen and Princeton) Coal Area:

Princeton Colliery.—This colliery, the first in the Central District to ship coal and for a long time the leading colliery, is now owned by the Princeton-British Columbia Colliery Company, Limited, successors to the Princeton Coal and Land Company.

The mine was originally developed by two slopes, which reached the coal at a distance of 200 feet from the point of commencement. These slopes were continued for about 4,000 feet on the seam, which dips uniformly at 10 degrees. The seam averages about 20 feet in thickness, but includes many clay bands, and only ten feet of its upper portion has been worked.

The method of working has been pillar-and-stall, coal cutting machines of the post type being used for mining the coal.

Trouble was experienced with spontaneous combustion, especially in the area where pillars had been taken out, and some years ago it was decided to extract what pillars were available, with the result that the mine has now been entirely abandoned. At present a shaft, sunk in the forks between the Similkameen and Tulameen rivers, is used for the production of coal.

Another company, the Tulameen Valley Coal Company, is also operating in this district, having opened a colliery about two miles west of Princeton, on the Tulameen river. The seam is similar to the one mined at Princeton colliery and the methods of working are the same. The coal is hauled by auto truck to Princeton, for disposal there or for shipment.

Coalmont Colliery.—This colliery is operated by the Coalmont Collieries, Limited, and is situated on the west bank of the North fork of Granite creek, about 4.5 miles from

the town of Coalmont on the Kettle Valley railway. The seam being worked here is in the Tulameen coal basin, situated about twelve miles northwest of Princeton. It averages between ten and twelve feet in thickness, and is rated as a semi-bituminous coal.

Two mines are operating at present. Tunnels have been driven in from the outcrop, and the mine developed by the pillar-and-stall method. The coal is mined by hand, and loaded in cars of one ton capacity. Haulage, ventilation, etc., are the same as in the other mines described.

The coal, as brought from the mine, is in a box which can be lifted from the running gear of the car and transferred to an aerial tramway, by which it is transported 2.5 miles to the tipple, situated convenient to the Kettle Valley railway and at an elevation 1,320 feet less than the upper tramway terminal.

The stationary cable on the empty or return side of the tramway is 1½ inches in diameter and on the loaded side 1½ inches, while the hauling cable is ½-inch. The tramway is equipped with friction brakes, and a 50 h.p. constant-speed motor, running at 640 r.p.m. and connected by gear to the return grip pulley, keeps the tramway running at a constant speed. The motor acts as a brake when the tramway is running, by generating power. The lower terminal, at the tipple, is provided with automatic detachers and attachers, the loaded cars running by gravity to the dump and emptying into the coal chute. Each car, or bucket, makes a return journey in 1.25 hours, and with a total of 80 buckets on the tram, the delivery of coal to the tipple is at the rate of 60 tons per hour.

The main power plant is situated at the tipple, and consists of two water tube boilers with a combined capacity of 500 h.p., which supply steam to a Corliss engine driving a three-phase, 480 ampere, 550 volt, generator. The electric current is stepped up to 10,000 volts and transmitted two and a half miles over high-tension cables to the mine, where it is reduced by transformers to 550 volts for working purposes and to 110 volts for lighting.

For the accommodation of employees, there is provided a rooming house with thirty-six rooms, a dining hall, wash and change rooms, etc., as well as a number of houses for the married employees.

Other Areas:

The only other areas in the Central District from which there has been any production of coal are Chu Chua, on the North Thompson river; Telkwa River, in the Omineca mining division, on the Canadian National Railways line to Prince Rupert; and Hat Creek, about 15 miles west of Ashcroft. In each case the operations have been rather in the nature of prospecting than actual mining, and the extent and value of the coal deposits remain to be proved.

Production from mines in the district during 1925, and other particulars of the industry, are given in the table below:

Operating Company	Production (long tons)	Number of employees	Production per employee per day	
Middlesboro Collieries, Ltd	37,686	130	1.62 tons	
	5,927	16	1.34 "	
	54	3	0.51 "	
	117,877	301	1.40 "	
	6,581	22	1.49 "	
	7,079	36	0.79 "	

GENERAL

The present markets for British Columbia coal are for railways and steam-boat consumption, gas and coke production, and general industrial and domestic purposes. The Western District supplies the greatest proportion of the steam-boat trade, both ocean-going and coasting. The Eastern District supplies practically all the coal used for coke making, except for a small portion shipped from the Granby Company's Cassidy colliery to their own smelter at Anyox, where it is treated in by-product ovens. All the districts provide coal for railways and local industrial purposes, and from both the Vancouver Island and Crowsnest fields shipments are made to points in the United States.

All the coal is used without any attempt being made to obtain any of its by-products, apart from coke, except in the case of the relatively small amount shipped to Anyox. In the Crowsnest Pass coalfield, which has supplied practically the whole of British Columbia's coke in recent years, ovens of the bee-hive type are still used, and as over five million tons of coal have been burned to produce coke, it is interesting to speculate on the value of the by-products that have been lost, and to wonder how long this is likely to continue.

The competition of fuel-oil, which has to a great extent retarded the development of the Vancouver Island field, should become less keen as the price of oil increases—and there are already predictions that the day of cheap fuel-oil is almost past. Even at the present time, many large manufacturing plants which have been using oil are reverting to coal, and as science develops more economical methods of using coal, the economic margin in favour of oil over coal should become less.

Modern research, in addition to broadening the field for the uses of coal and its products, can also assist by devising more economical methods of production, and in this way make it possible for coal to compete successfully with other sources of power and heat. The extraction of oil from shale, as a successor to the oil well, should not prove a very serious competitor to coal, as it has been estimated that it will require the mining and treating of about 10,000 tons of shale to produce sufficient oil to drive a modern battleship for a single hour. On the other hand, the production of oil from coal may well prove the solution of our coal mine development problem, and the process recently invented by Dr. Bergius, of Germany, which promises to produce gasoline, kerosene, and lubricating oil from coal, and having a value of from twenty to twenty-five times the market value of the coal used, marks a long step in the progress along this line.

There is room for a great deal of improvement in the coal industry of British Columbia, both along practical mining and scientific lines, and on this will depend the successful development of the industrial life, the progress and prosperity, of a great part of the Province.

MINE REGULATIONS

In British Columbia, the title to coal lands is derived from the Government of British Columbia, except those lands situated inside the Railway Belt, Peace River block, and certain Dominion reserves in the Crowsnest coalfield.

Chapter 162 of the Revised Statutes of British Columbia contains regulations covering application for a license to prospect for coal, locating of coal lands, and application to purchase same. Under its provisions, every piece of land sought to be obtained under a prospecting license shall be rectangular in shape and shall include within the general limits therein defined land not exceeding six hundred and forty acres. A block of this maximum size shall measure eighty chains by eighty chains, and all lines shall run true north and south, and true east and west.

Chapter 30 of the Revised Statutes, "Mineral Right-of-Way Act", makes provision for any holder of either mineral or coal mining claims to secure a right-of-way, over, under, or through any lands necessary for the development and working of the mining property.

Chapter 171 of the Revised Statutes, "Coal Mines Regulation Act", covers the operation of the coal mines in the Province. Various terms are first defined, so as to make clear the scope of application of the several sections, of which there are thirteen, covering:

- 1. Regulation of employment and wages.
- 2. Mining shafts, outlets, sub-marine coal areas, and division of mines.
- 3. Employment of managers, overseers, and coal-miners.
- 4. Examinations and enquiries as to competency.
- 5. Returns and notices.
- 6. Protection of abandoned mines.
- 7. Inspection of mines.
- 8. Plans of mines.
- 9. Arbitrations, to settle disputes as to the Act.
- 10. Inquests as to cause of fatal accidents.
- 11. Rules as to ventilation, blasting, etc.
- 12. Mine rescue work and apparatus.
- 13. Supplemental, for prosecutions and penalties under the Act.

The Rules governing the use of electric power in coal mines have already been referred to, and there are in addition special regulations for precautions against coal dust.

Under the Coal Mines Regulation Act, every mine must be under the control and daily supervision of a manager, who must be the holder of a certificate of competency granted under the Act. The underground working of every mine shall be under the daily charge of an overman or overmen, shift-boss, fireboss, and shot-lighter, all of whom must hold certificates granted under the Act to cover the duties assigned them.

No person shall be employed as a coal-miner in any mine, who is not in possession of a certificate of competency as such; in other words, every person employed underground at the mine, except labourers, is required to hold a certificate of competency.

The Department of Mines comes under the charge of a Minister under the Crown, with a Deputy Minister. The work of inspection under the Coal Mines Regulation Act is covered by a body of Inspectors of Mines appointed by the Lieutenant-Governor in Council, subject to the provisions of the Civil Service Act. The Minister of Mines may designate one of the Inspectors of Mines as Chief Inspector, assign him his duties and designate such place as the office of the Chief Inspector as he may see fit.

Every Inspector of Mines must hold a mine manager's certificate, and any person who practises or acts, or is a partner of any person who practises or acts, as a land agent or mining engineer, or as a manager, viewer, agent, or valuer of mines, or arbitrator in any difference arising between owners, agents, or managers of mines, or is otherwise employed in or about any mine (whether the mine is one to which this Act applies or not) shall not act as an Inspector under the Act, and no Inspector shall be a partner or have any interest, directly or indirectly, in any mine in the district under his charge. Provision is also made under the Act for the removal of the Inspector for cause. The owner, agent, or manager must keep a plan of the mine, drawn to a scale of not less than one hundred feet to the inch, showing the workings up to a date not more

than three months previous, and a copy of this plan must be posted in some conspicuous place, at or near the main entrance to the mine, and all roads used as a means of egress shall be conspicuously marked on this plan.

Provision is also made for arbitration, in the event of any difference of opinion arising between the coal company and the Mines Department.

A fairly wide range of General Rules are provided under the Act, a copy of which must be kept posted near the mine entrance, covering ventilation, blasting, lighting underground, inspections by firebosses and other officials, machinery, and report books. In these General Rules, provision is made for inspections on behalf of the workmen, sanitary provisions underground and on the surface, and the maintainance of ambulance supplies.

Special Rules must be drawn up, with special reference to the colliery, subject to the approval of the Minister of Mines, and when approved these must be kept posted in a similar manner to the General Rules.

Each colliery is required to maintain a certain amount of mine rescue apparatus, the number of such to be approved by the Minister of Mines, and it is incumbent on the owner, agent, or manager to have all certificated officials who are physically fit, and not less than three per cent or such number as the Chief Inspector may deem sufficient, of the workmen, trained in the use of such established mine rescue apparatus.

Provision is also made in the Act to cover penalties for either evasion or breach of its provisions, or the provisions of the Special Rules.

All certificates of competency under the Mines Act are granted by a Board of Examiners, appointed for that purpose, and of which the Chief Inspector is chairman.

The death rate from accidents has, in the past, been very heavy, but the new rules covering mine hazards, and better enforcement of existing rules, have reduced accidents during the past few years. During the year 1925, the fatal accident rate per thousand employees was 1.10 as compared with 4 per

thousand for the ten-year period. The rate of fatal accidents for the year 1925 per million tons of coal raised was 2.45, as compared with 9.3 for the previous ten-year period.

In concluding, I wish to thank the officials of the several coal companies, who have greatly assisted in the preparation of this paper by supplying data and photographs, and also by making many helpful suggestions; and in these respects I am particularly indebted to Mr. Hartley Wilson, general manager of the Crow's Nest Coal Company.

SECTION 2.—THE COAL MINING INDUSTRY OF ALBERTA

By W. J. DICK (Member, C. Inst. M. & M.)*

(Jasper, Alta., Meeting, September 19th, 1927)

THE NATIONAL IMPORTANCE OF COAL

Day by day it becomes more evident that coal is the mainspring of modern material civilization. In addition to its use as a source of motive power and heat, it has of late years been found to yield by-products of inestimable value, supplying gas for heating, lighting, and industrial purposes; products forming the basis of the aniline-dye industry; and various chemical products for use as fertilizers, disinfectants, drugs, and explosives. Its importance to a nation at war has been well exemplified in the recent Great War.

As regards the supremacy of coal as a source of heat and power, and the impossibility of finding a substitute, Professor Tyndall states:

"I see no prospect of any substitute being found for coal, as a source of motive power. We have, it is true, our winds and streams and tides; and we have the beams of the sun. But these are common to all the world. We cannot make head against a nation which, in addition to those resources of power, possesses the power of coal.

"It is no new thing for me to affirm, in my public lectures, that the destiny of this nation [Great Britain] is not in the hands of its statesmen, but in those of its coal owners; and that, while the orators of St. Stephen's are unconscious of the fact, the very life-blood of this country is flowing away".

The value of coal has, therefore, not yet been fully realized, and the progress of science, and improvements in the arts, will tend to increase the supremacy of steam and coal. While hydro-electric energy will replace it to a certain extent in Canada, the dependence of numerous industries upon coal and coal products for their raw materials, ensures that coal will always be required, and on a larger and increasing scale. Upon the exhaustion of the oil-wells, all the petroleum products which are now obtained from them must, as far as possible, be obtained from products distilled from coal.

^{*} General Manager, Cadomin Coal Company, Limited.

COAL RESOURCES OF ALBERTA

It is natural, perhaps, that there should be some confusion. even in the minds of some of our own citizens in Alberta, respecting the coal resources of the Province. They are potentially so large, and our actual knowledge of them is so slight, that there is considerable scope for a difference of opinion regarding any estimate that may be made of them.

In 1912, the International Geological Congress undertook to secure estimates of coal resources from all countries, in order that we might have what might be called a stock-book showing the coal resources of the world (1). In the report that was published, the coal resources were considered in two groups:

Group 1.—Including seams of one foot and over in thickness to a depth of 4,000 feet.

Group 2.—Including seams of two feet and over in thickness at depths between 4,000 feet and 6,000 feet.

The total coal resources of Alberta, of Canada, and of the world were given as follows:

(In million tons)

	Alberta	Canada	World	Alberta per cent of Canada	Alberta per cent of World
Anthracite Bituminous coal.	768 198,092	2,158 283,661	496,846 3,902,944	35.59 69.83	.15 5.07
Sub-bituminous and lignite coals	876,179	948,450	2,997,763	92.38	29.23
Totals	1,075,039	1,234,269	7,397,553	87.10	14.53

In other words, on the basis or formula applied by the International Geological Congress in the preparation of their report, Alberta was credited with 14.53 per cent of the total coal resources of the world, and about eighty-seven per cent of the total coal resources of Canada.

⁽¹⁾ The Coal Resources of the World. In three volumes. Morang & Co., Ltd., Toronto, Ont.

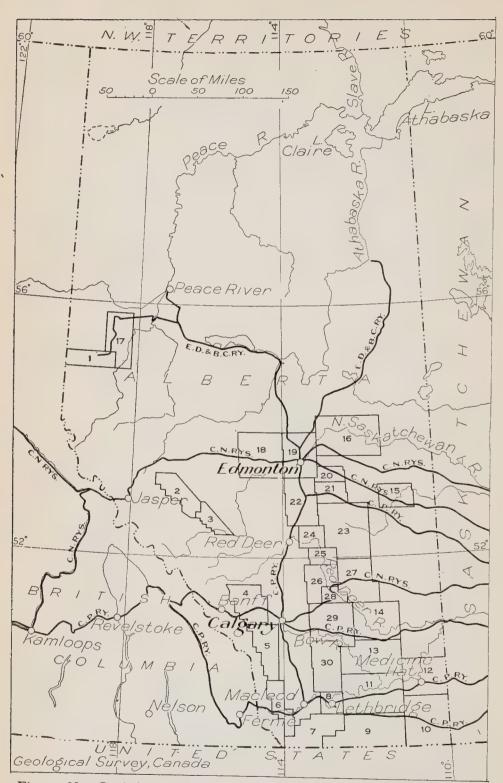


Figure 28.—Coal Areas of Alberta, as outlined by the Scientific and Industrial Research Council of the Province of Alberta.

The estimate for Alberta is based on the assumption that there are, in all, 81,878 square miles underlain by coal. Practically the whole of Alberta's estimate comes in Group 1, i.e., seams of one foot and over to a depth of four thousand feet.

Since 1913, much new information has been gained, both through geological investigation and in the course of mining the coal, but it still remains impossible to give anything like a complete estimate of the coal resources of the Province. Apart from the academic interest that such knowledge might possess, it could make very little difference to our future, as our known reserves are out of all proportion to our production, either present or probable in the immediate future.

The Report of the Alberta Coal Commission, 1925, commenting on this, states:

"In evidence before the Commission, Dr. Allan gave an area of 12,000 miles as the total on which they were now inclined to think coal existed; although, as will be seen, his actual estimate prepared for the Commission deals with only about one-fifth of even this restricted territory. Since the bulk of the world's coal in the estimates quoted occurs in the United States and Europe, where conditions are better known, and moreover consists of the more uniformly occurring Carboniferous coals, it is hardly possible to expect a shrinkage of all the estimates. This Commission, therefore, believes that even on the definition of seams of one foot and over to a depth of 4,000 feet, the present state of knowledge demands a reduction of the Alberta estimates, both as to area and quantity, to at most one-fifth of the 1913 calculations. Even allowing something for similar reductions elsewhere, it will more nearly represent the latest information to say that Alberta probably has three per cent, and not fourteen per cent, of the world's coal resources. Certainly there appears to be no warrant for continuing to make the statement that Alberta has one-seventh of the known coal deposits".

The following is an estimate of reserves in some coal areas in Alberta, as prepared by Dr. Allan for the Alberta Coal Commission. This estimate is based on coal seams two feet or over in thickness and within 1,000 feet of the surface. Note omission of any estimate for Clearwater, Panther, Smoky River, and Prairie Creek coal areas.

Coal Reserves of Alberta (Allan)

D	ivision and Coal Area	Area	Tons of	Total of Division,
No.	Name	(sq. miles)	2,000 lb.	tons of 2,000 lb.
1.	Bituminous coal:			
1.	Crowsnest:			
	K.7 Oldman	72	2,488,320,000	
	K.10 Crowsnest	216	6,220,800,000	8,709,120,000
2.	Canmore:		, , , , , , , , , , , , , , , , , , , ,	
	K.7 Cascade	144	6,451,200,000	
	K.8 Highwood	108	8,294,400,000	14,745,600,000
3.	Brazeau:		.,,, ., .	
	K.4 Nordegg	36	1,119,744,000	
	K.5 Clearwater			
	K.6 Panther			1,119,744,000
4.	Mountain Park:			
	K.1 Smoky River			
	K.2 Brule	36	1,866,240,000	
			, , ,	
	K.4 Mount. Park	108	4,976,640,000	6,842,880,000
,	Total bituminous	720		31,417,344,000
2. \$	Sub-bituminous coal:			
1.				
1.	B.7 Pincher	26	202 244 000	000 044 000
_		36	303,344,000	303,344,000
2.	Canmore:	4.0		
	B.5 Morley	108	1,468,800,000	
	B.6 Pekisko	108	2,176,000,000	3,644,800,000
3.	Brazeau:			
	B.4 Saunders	72	1,566,720,000	1,566,720,000
4.	Mountain Park:			
	B.2 Prairie Creek			
	B.3 Coalspur	144	6,092,800,000	6,092,800,000
5.	Lethbridge:		, , , , , , , , , , , , , , , , , , , ,	,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,,
	B.9 Lethbridge	216	1,175,040,000	
	B.8 Magrath	72	626,688,000	1,801,728,000
13.	Peace River:	12	020,000,000	1,001,720,000
10,	B.1 Halcourt	72	1 175 040 000	1 175 040 000
	A STATE OF THE STA	14	1,175,040,000	1,175,040,000

Coal Reserves of Alberta (Allan)—Continued.

Divi No.	ision and Coal Area Name	Area (sq. miles)	Tons of 2,000 lb.	Total of Division, tons of 2,000 lb.
3. Li	gnite coal:			
5.	Lethbridge:			
	B.12 Taber	114	614,400,000	
	B.10 Milk River	112	1,024,000,000	1,638,400,000
6.	Medicine Hat:			
0,	B.16 Empress	1	5,120,000	
	B.13 Redcliff	1	6,144,000	
	B.11 Pakowki	41	419,840,000	431,104,000
7.	Brooks:			
•	E.11 Gleichen	4	61,440,000	
	E.12 Champion	10	153,600,000	
	B.15 Steveville	3	15,360,000	
	B.14 Brooks	36	184,320,000	414,720,000
8.	Drumheller:			
	E.7 Big Valley	7	215,040,000	
	E.8 Carbon	33	709,632,000	
	E.10 Drumheller	288	3,072,000,000	
	E.9 Sheerness	17	261,120,000	4,257,792,000
9.	Ardley:			
	E.6 Ardley	15	537,600,000	
	E.5 Castor	33	460,800,000	998,400,000
10.	Pembina:			
	E.14 Whitecourt		30,720,000	1 070 000 000
	E.1 Pembina	72	1,843,200,000	1,873,920,000
11.	Edmonton:		= 100 000	
	B.19 Rochester		5,120,000	1 207 520 000
	E.2 Edmonton	115	1,382,400,000	1,387,520,000
12.	Tofield:		40004000	
	B.18 Pakan	1	16,384,000	
	E.3 Tofield		307,200,000 107,520,000	
	E.4 Camrose		30,720,000	
	E.13 Wetaskiwin		16,384,000	478,208,000
	B.17 Wainwright	4	10,504,000	1,0,200,000
13.	Peace River:		20,480,000	20,480,000
	B.20 Sexsmith	. 2	20,460,000	20,400,000
	Total lignite	940		11,500,544,000
,	Total all coals	2,488		57,512,320,000

Summary

Kind of coal	Square miles	Av. aggreg. thickness of coal seams	Tons per acre for thickness of one foot	Fetimated tone	Taken at 50 per cent recoverable
Bituminous Sub-bituminous Lignite Total	720 828 940 2,488	38 ft. 16 ft. 12 ft.	1,800 1,700 1,600	31,417,344,000 14,594,432,000 11,500,544,000 57,512,320,000	15,708,672,000 7,297,216,000 5,750,272,000 28,756,160,000

It is a well known fact that a coal seam two feet in thickness cannot *now* be considered a workable seam, but in a consideration of total coal reserves, and in view of the experience of other countries, we are justified in including all seams over two feet in thickness. The total production of coal from Alberta mines from 1886 to the end of 1926, has been in the neighbourhood of ninety-four million tons. This is about three-tenths of one per cent of the above workable reserves as estimated by Dr. Allan.

The known partial reserves are capable of sustaining our maximum production in any past period for about four thousand years, so that we may safely let the matter of actual reserves rest for future geological exploration to determine.

CLASSIFICATION OF ALBERTA COALS

It is quite easy to classify the higher-grade fuels of Alberta, such as the semi-anthracite and bituminous coals; but coals ranking below these grades, and having a widespread distribution in Alberta, are difficult to classify without forming unfair prejudices in the mind of the public respecting certain of these coals that may fall within a general classification.

The coals of Alberta have been classified in descending order as follows:

- 1. Semi-anthracite
- 2. Bituminous
- 3. Sub-bituminous
- 4. Lignite

The writer is of the opinion that there is no true lignite mined in Alberta, and that the classification of certain of our coals as lignite is unfair, as those so classed are excellent coals of comparatively high fuel value.

The sub-bituminous coals in the classification above are also good domestic coals, as well as good steaming coals, so that the classification might better be arranged as follows:

- 1. Semi-anthracite
- 2. Bituminous
- 3. Sub-bituminous:
 - (a) High-grade domestic coals and steam coals
 - (b) Domestic coals

A classification (1) suggested by Mr. Edgar Stansfield, Secretary and Chemical Engineer of the Scientific and Industrial Research Council of the Province of Alberta, is reproduced in the table (insert). This classification is for coals below the anthracite, or, as commonly called, the semi-anthracite class.

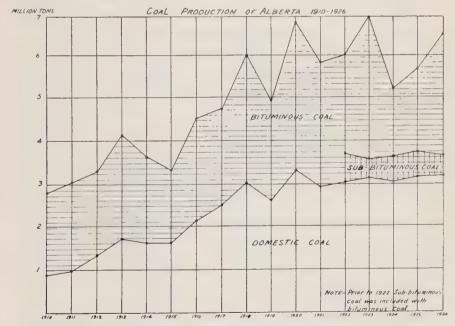


Figure 29.—Coal Production of Alberta, 1910-1926.

^{(1) &}quot;A Chemical Survey of Alberta Coals", by Edgar Stansfield; Trans. Can. Inst. Min. & Met., Vol. XXVIII, 1925, p. 266.

OUTPUT OF COAL FROM ALBERTA MINES

Coal mining in what is now the Province of Alberta was begun at Medicine Hat in 1883. In 1896 mines were opened at Lethbridge, and the total output for that year was a little over 43,000 tons. Following this, some of the earliest mines were opened at Canmore, Edmonton, and Crowsnest Pass. There was no rapid growth in production until 1900, when the output was 311,500 tons. By 1910 it had jumped to over two and three-quarter million tons, or an increase of over eight hundred per cent, and since that time it has more than doubled.

The following table gives the output in gross tons for each year from 1910 to 1926, inclusive (See also Figure 29):

Year	Domestic	Bituminous*	Total
1910	878,011	1,896,961	2,774,972
1911	964,700	649,745	1,641,445
1912	1,341,389	1,926,371	3,267,760
1913	1,763,225	2,374,401	4,137,626
1914	1,679,401	1,953,367	3,632,768
1915	1,682,922	1,626,237	3,309,159
1916	2,172,801	2,335,259	4,508,060
1917	2,537,828	2,206,868	4,744,696
1918	3,035,061	2,982,334	6,017,395
1919	2,611,009	2,325,787	4,936,796
1920	3,359,308	3,419,021	6,778,329
1921	2,943,141	2,897,380	5,840,521
1922	3,086,569	2,923,426	6,010,095
1923	3,161,741	3,744,820	6,906,561
1924	3,096,660	2,107,053	5,203,713
1925	3,156,359	2,727,035	5,883,394
1926	3,160,029	3,348,879	6,508,908

^{*}Includes sub-bituminous.

From present indications, it is evident that the coalmining development will not increase so rapidly in the future as it has in the past, unless additional markets can be secured. The fact of the matter is that what we require is, not a further development of our coal areas, but a market for the tonnage that can be produced from the mines already equipped and developed for production.

GEOLOGICAL HORIZONS OF ALBERTA COALS (1)

When discussing the quality of the different coals in Alberta, it is important to note that there are three coal-bearing horizons within this Province, each belonging to a different geological age, and separated from each other by formations from 700 to 3,000 feet in thickness.

The coals from these various horizons differ in grade and quality. Those from the lower and therefore older formations. on account of their age and of the greater weight to which they are subjected by the overlying formations, are of a harder and better quality, and more suitable for steam and coking purposes. The older coals are lower in moisture, higher in fixed carbon content, and, as a rule, higher in volatile constituents. On the other hand, the grade of coal in a single horizon improves towards the foothills and mountains. This is due to the fact that towards the west the coal seams have been more intensely compressed and otherwise carbonized by the stresses from the mountain-building forces, which were uplifting the present Rocky Mountain system. These stresses and the formation of the coal in Alberta were in no way connected with volcanic forces, as is so commonly believed to be the case. The dynamic forces which assisted in compressing and metamorphosing the coal seams in Alberta have been those connected with orogenic or mountain-building movements, or with much broader continent-wide uplift. This latter movement has been the cause of raising the coal seams and the associated beds from approximately sea level, where they were formed, to their present elevation of over 2,200 feet above sea level.

The variation in the age of the coal, and also the distance from the mountains, has produced all grades of coal in Alberta, from a medium quality of lignite under the plains, to an anthracite within the front ranges of the Rocky mountains.

⁽¹⁾ From "First Annual Report on the Mineral Resources of Alberta", by John A. Allan.

Three coal horizons which are worked in Alberta occur in the following formations, from the youngest to the oldest:

1. Edmonton formation (Uppermost Cretaceous).

2. Belly River formation (Middle Part of Upper Cretaceous).

3. Kootenay formation (Lower Cretaceous).

In certain localities in Alberta and in Saskatchewan there are workable seams in the beds of lower Tertiary age. This coal is all lignite in character.

The Kootenay coal measures are exposed and worked principally within the front range of the Rocky mountains, or in the foothills close to the front escarpment of the mountains. The coal varies from bituminous to anthracite. The most important bituminous coal basins in these measures are Crowsnest; Livingstone and Moose Mountain, which include deposits on the upper part of Sheep river; Bow Valley, at Bankhead and Canmore; Brazeau, Big Horn, Mountain Park, Brule Lake and Pocahontas, and Smoky River.

The coal seams in the Belly River formation are worked in the Lethbridge basin.

Other exposures occur in the vicnity of Medicine Hat; the lower part of the Red Deer river; South Saskatchewan and North Saskatchewan rivers, where these streams cut through the Belly River formation. The area in Alberta underlain by the Belly River coal horizon is very much larger than the actual outcrops would indicate. At Edmonton the Belly River coal seam has been located, by drilling, at 1,400 feet, and at Tofield the corresponding seam occurs at 1,050 feet.

In the vicinity of Calgary, where there is a considerable thickness of Tertiary rocks capping the Cretaceous formation, three coal seams have been drilled through in what is believed to be the Belly River formation. The depths given for these seams are as follows:

A 5-foot seam at 2,562 feet,

A 7-foot seam at 2,655 feet, and

A 4-foot seam at 2,875 feet,

below the surface. For the time being, these seams are too deep to be worked profitably so long as equally suitable coal can be obtained at, or close to, the surface.

The coal from the Belly River measures ranges from bituminous to sub-bituminous. Dowling estimates that these coal measures are distributed over 25,974 square miles, and contain an actual and probable reserve of 189,450 million tons of coal.

The Edmonton formation contains two coal horizons. The uppermost coal seam occurs near the top of the formation and varies in thickness from about 5 feet, south of the Bow river, to a maximum of 25 feet on the North Saskatchewan west of Edmonton. This upper seam is worked at a number of localities towards the foothills of the mountains.

About 500 to 600 feet below this upper seam, a number of seams occur near the base of the formation. These seams are known to extend from near the International Boundary line northward to the vicinity of Morinville, north of Edmonton. These seams are worked at a number of localities which include Drumheller, Tofield, Edmonton, Clover Bar, Sturgeon Valley, and Morinville.

It has been estimated by Dowling that the actual total area containing available coal in the whole Edmonton formation is 25,235 square miles, which will produce 383,697 million metric tons. To this amount may be added a probable reserve in this formation covering 27,170 square miles which would produce 417,261 million metric tons.

The quality of the coal in this formation ranges from bituminous down to lignite. Much of the coal of this age may be classed as sub-bituminous. The term 'domestic coal' has been applied to the grade of material mined from this horizon.

The following analyses indicate the range in composition and calorific value of coals from each of coal-bearing horizons:

	Kootenay	Belly River	Edmonton	Paskapoo
Fixed carbon Volatile matter. Moisture Ash	55–80% 12–30% 0.6–2.5% 10–20%	4 7– 56% 25–34% 5–9.5% 8–15%	43–47% 25–34% 15–24% 7–15%	38–43% 28–35% 23–27% 6–11%
B.t.u. per lb	11,500–14,500	10,000-12,000	8,500–10,500	8,000–9,500

The wide range of volatile matter and fixed carbon in the coals of the Kootenay formation is due to the fact that the normal bituminous coal has been changed to semi-anthracite coal, due to pressure.

KOOTENAY FORMATION

Generally speaking, the coal seams of this formation have steep dips and outcrop in a hilly or mountainous region. As the railways, which afford transportation facilities, follow the valleys, the usual practice is to develop the mines by tunnels driven from the level of the valleys, the coal being mined to the rise of these tunnels.

These conditions, where the cover is not very great, are favourable to economical mining, as haulage, pumping, and ventilation do not present serious difficulties; but as soon as the coal to the rise has been mined and a second lift is undertaken, the conditions are less favourable. Owing to the steep dip, the cover over the workings quickly increases, and timbering, ventilation, pumping, and haulage problems become serious factors. Some of the mines are now mining a second lift; and, eventually, all will be working highly pitching seams under great cover, in which, also, there are a number of coal seams.

The mines are, as a rule, gaseous and dry, and great care is exercised in providing adequate safety measures.

Mining of seams in the Kootenay formation is carried on in the following districts in Alberta:

Crowsnest Pass district. Banff-Canmore district. Brazeau district. Mountain Park district. Jasper Park district.

Crowsnest Pass District (Alberta)

This district embraces that area contiguous to the Crow's Nest Pass branch of the Canadian Pacific railway and extending eastward from the inter-provincial boundary to Burmis.

Geology.—The Fernie shales (Jurassic) and Cretaceous coal measures lie conformably on the Devono-Carboniferous During Laramie (Upper Cretaceous) time the limestone series was tilted or folded into its present position, forming the Rocky mountains. The limestone ranges in this district comprise the main range of the Rockies and the Livingstone range, the latter rising some fifteen miles to the east, where the limestone is exposed on the railway at Frank. Laramide revolution was brought about by a great thrust from the west which caused the folding or tilting of the heavy limestones and of the overlying rocks. This has resulted in a repetition of the coal measures, so that they now form five or six bands, having northerly strike, in each of which coal mining may be undertaken. West of the Livingstone range the coal measures dip to the west. East of the Livingstone range, which is caused by a large overturned fold faulted on the east, the dip of the coal tends towards the east. Exceptions to this general statement are the Hillcrest area, which is in the nature of a basin having on the east a westerly dip and on

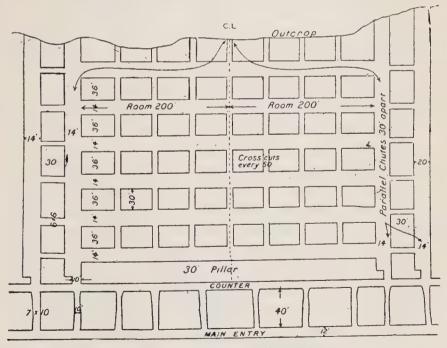


Figure 30.—Across-the-pitch mining.

Sketch showing all the rooms driven between two pairs of chutes. Pillars are removed by commencing at the point indicated by arrows.

the west a steeper easterly dip; and at Bellevue, where the measures are folded into synclines and anticlines possessing east and west dips.

The principal mining centres are at Coleman, Blairmore, Bellevue, and Hillcrest, towns on the Crow's Nest branch of the Canadian Pacific railway. The railway is located in the valley and its general direction is at right angles to the measures. It thus affords comparatively easy access to the coal seams, and it has greatly aided in their development.

General Mining Methods.—The average dip of the coal seams is about thirty-two degrees. In general they are opened up from the railway level, either by tunnels driven on the seam along the strike of the coal, or by level rock cross-cuts to the coal. The coal is first mined to the rise from the tunnel level. The coal to the dip is developed by slopes in the coal, with levels varying from three hundred to seven hundred feet apart, measured along the pitch of the seams. Where two or more workable seams are close together, they are developed by rock cross-cuts from a seam already developed.

The system of mining adopted is pillar-and-stall. Two variations of this system are in use, stalls up the pitch and stalls across the pitch (Figure 30). The former is the general practice, in which the stalls are chutes driven up the full pitch and the coal slides by gravity to the main level below.

All the mines in this district have adopted a systematic and uniform method of mining, and the total extraction is high.

The West Canadian Collieries, Ltd.:

The West Canadian Collieries, Limited, was incorporated in 1902, under English law, and is one of the large pioneer operators in this district, in which it has developed four mines—the Lille, Blairmore, Bellevue, and Greenhill mines—but only the two last named are now operating.

Lille Mine.—The first mine of West Canadian Collieries was opened in 1901 at Lille, five miles north of Frank, on the Crow's Nest Pass branch of the Canadian Pacific railway.

The seam mined (No. 1 seam) is 4 ft. to 6 ft. thick and dips at 30 to 40 degrees to the west. The system of mining was across the pitch, chutes being run up the pitch every 200 feet, and rooms broken off on the level. The coal was trammed

to the chutes and discharged to the gangway. When panels were completed, pillars were taken out, receding from one chute to the next. The recovery was ninety per cent.

For haulage, compressed-air locomotives, with air at 1,000 lb. pressure, were used. The pit cars, capacity $1\frac{1}{2}$ tons, were loaded from the chutes and taken to the tipple in trips of 30 to 35 cars each. The mine was ventilated by means of a 6-foot steam-operated Murphy fan, which furnished 45,000 feet of air per minute. The general mine equipment included tipple, with Phillips over-dump shaker screens, picking belt, etc., and made three products: mine run coal, which was supplied to the railways; screened coal, for domestic fuel; and slack, which was used for coke making.

The boiler house was equipped with four 150 h.p. return tubular boilers, feed pumps, etc.; and the power house contained a high-pressure air compressor, having a maximum of 1,000 lb. per sq. in. and a capacity of 776 cubic feet of air per minute.

Fifty Bernard regenerative non-recuperative coke ovens were operated. In these, the gases passed off through side walls and thence under floor to stacks. Eight tons of coal were charged to each oven, producing six tons of coke. The time of coking was 48 hours, so that each day 25 ovens were discharged, making a total of 150 tons of coke.

The slack from the mine was taken to the wet washery, equipped with eight Luhrig jigs. From here the washed slack was elevated to drying bins, and thence to the coke ovens.

Five return tubular boilers, using waste from the washery, supplied power to operate the washery, electric lighting plant, coke-oven elevators, ram, etc., and during the winter the surplus steam was used for heating the washery.

Adjacent to the mine were seventy-five frame cottages for employees, a superintendent's house, doctor's house, hospital, school, and stores—all with electric light and running water—as well as a water-works for fire protection.

A branch line, owned by the Company and known as the Frank-Grassy Mountain railway, connected the mine with the main line of the Canadian Pacific railway. It was of standard gauge, and, including sidings, had a length of 6.95 miles.

The Lille mine had a daily output of 500 tons of mine run and screened coal, and 150 tons of coke. It was closed down early in 1914, mainly on account of the collapse of the coke market.

Blairmore Mine.—This mine, situated at Blairmore, on the south side of the main line of the Crow's Nest Pass branch of the Canadian Pacific railway, was opened in 1909.

The seam worked (No. 2 seam) is 12 feet thick, with dip 45 degrees and average lift 1,000 feet. The system of working was pillar-and-stall, with rooms 12 feet to 15 feet wide, and pillars 30 feet. Rooms were connected by cross-cuts.

Haulage was on 40-lb. rails in two-ton capacity pit cars, three horses hauling twenty-five cars per trip. The mine had a capacity of 700 tons per eight-hour shift, and the principal product was mine run coal.

A six-foot reversible-hood Murphy fan furnished 40,000 cu. ft. of air per minute. Other equipment included a boiler house, with two return tubular boilers, of 150 h.p. and 125 lb. working pressure; power house, with one 18 by 18 Robb engine, and two alternators, one 100 k.v.a. and the other 75 k.v.a., three-phase at 2,300 volts, furnishing power for the fan and for mine lighting; and tipple, car haul, Phillips automatic cross-over dump bar screens, etc.

The Blairmore mine is at present closed down.

Bellevue Mine.—This mine is situated four miles east of Blairmore. It was opened in 1903 and is working the southerly extension of the Lille seams, which at this point are both thicker and of greater extent. There are six coal seams, but at present work is confined almost entirely to one of these, the No. 1, which has a thickness of 12 feet. The other seams, numbered from 2 to 6, are, in order, 12, 3, 6, $3\frac{1}{3}$, and 2 feet thick. The dip is thirty degrees to the west and the lift about 200 feet to 1,400 feet above the gangway.

The system of working is room-and-pillar, a gangway being driven about every 500 feet of lift. Rooms 20 feet wide are driven up the pitch at fifty-foot centres, and connected with cross-cuts. When development is sufficiently advanced, pillars are extracted, commencing at the airway or gangway immediately above and working down the pitch.

Eight levels are driven on the strike, No. 6 level being the main gangway at railway-track level. Five levels above this, on the lift, two slopes are turned off the main gangway in the coal at a pitch of twenty to twenty-four degrees to work the lower levels. The panel system and top airway system are used throughout, ten rooms to the panel.

In the slope workings, the coal is dug with French F.A.M. pick machines. Each machine weighs about 14 pounds and is actuated by compressed air. Two men are employed in each twenty-foot room, each man operating his own machine. The advantages of these machines are elimination of explosives,

coarser coal, and often better wages to employees.

A Jeffrey shot-wall coal cutting machine is being experimented with in the main gangway to expedite development

work.

Edison electric permissible safety lamps are used by the miners, while fire-bosses are furnished with Wolf safety lamps. Readings with the Birrell gas detector are taken daily.

No. 7 slope is equipped with a 4-in. centrifugal four-stage pump, operating against 470 feet head capacity, 500 gallons per minute. No. 8 slope has one 4-in. five-stage pump, with a

capacity of 250 gallons per minute.

Haulage is by compressed air locomotives throughout the mine, including slopes and water levels. A fleet of ten locomotives, two simple and eight compound, convey the coal from the chutes and working faces to the tipple. These engines are operated at 900 pounds working pressure. Cars are gathered at the main parting and made into trips of forty-five for haulage outside.

From No. 7 slope, the coal is hoisted by a single-drum Ingersoll-Rand hoist, at a speed of 900 feet per minute. This hoist is driven by a 300 h.p., 3-phase, 60-cycle motor, equipped with automatic control and air brakes, also magnetic brake. Hoisting from No. 8 slope is by a single-drum hoist, made in the Company's shops at Blairmore, driven by a 450 h.p. motor at a speed of 1,200 feet per minute. It also is equipped with automatic control and air brakes.

Air for this haulage and for the compressed-air picks and coal-cutting machines is furnished by two vertical Bellis and Morcom four-stage compressors, driven by two 520 h.p.

synchronous motors at 2,300 volts. These machines each make 1,200 cubic feet of free air per minute to 1,200 pounds pressure; 2,500 gallons of cooling water per hour per machine is pumped from the mine to a concrete tank, where it is filtered before being used in the compressors.

The principal method of ventilation is double airway, each panel having a separate split which in turn runs to the main return. Rock tunnels are run to No. 2 seam, where the main return is established completely independent of the working seam.

Two Keith fans, each with a capacity of 80,000 cubic feet per minute, and one Jeffrey fan, of 90,000 cubic feet per minute, take care of the total ventilation. All are of the exhausting type and are electrically driven.

In all dry portions of the mine, sprays are installed to keep airways and gangways in a damp state.

Mine equipment consists of a steel tipple, equipped with rotary dump with reciprocating feed to screens and three picking belts, each carrying a separate size of coal. Coal from $\frac{3}{4}$ -in. to $\frac{1}{2}$ -in. is carried by a separate Marcus table to a cleaning plant equipped with two pneumo-gravity cleaning tables, from which it is returned to the main railway car loading chute.

Ninety per cent of the coal is loaded in box-cars, which are loaded with a Christie box-car belt loader.

The boiler house and engine room contains twelve 150 h.p. return tubular boilers, working at 125 pounds pressure. These provide steam for the generators and two four-stage Ingersoll-Rand air compressors, auxiliary pumps, etc.

The mine offices and lamp house are housed in one large concrete and hollow-tile building, the lamp house being equipped with the necessary rectifiers, motors, charging racks for charging the batteries of the Edison electric lamps after each shift, etc.

A commodious wash-house constructed of concrete and hollow-tile is provided for the employees. It is equipped with individual steel lockers and has shower rooms furnished with hot and cold water for both the men and officials. It also contains a fully equipped first-aid room.

Other mine buildings include stores, stables, and machine shop, with motor pits and blacksmith shop.

The output of the mine, which has a capacity of 2,400 tons per eight hour shift, is practically all absorbed by the Canadian Pacific railway.

Greenhill Mine.—This mine was opened up in the autumn of 1914. It is situated to the north of the town of Blairmore on the main line of the Crow's Nest Pass branch of the Canadian Pacific railway, by which the mine is served.

There are two coal seams, No. 1 and No. 2, but the former only is being worked at present. The seam pitches to the west at about thirty-five degrees and the lift ranges up to 1,200 feet, above the main, or No. 5, level.

The mine is worked on the angle system and the present development consists of four levels above the railway tracks. Angles are turned off these levels in pairs at about 350 feet intervals, and back angles are broken off every fifty feet.

One slope, 1,200 feet long, is turned off the main, or No. 5, level in the coal, the vertical distance between the No. 6 level and No. 5 level being about 550 feet.

On this lower level, the panel and double airway system is used, modified by the rooms angling half way across the pitch and so reducing it from 35 degrees to 28 degrees.

In both these levels, pneumatic pick hammers are used exclusively, as at Bellevue mine and for the same reasons.

The haulage is main and tail rope, 8,800 feet long, operated by an Ingersoll-Rand double-drum hoist driven by a 300 h.p. motor.

The coal from the working faces, some 6,000 feet inside the main parting, is at present being handled by horse haulage, but the tonnage thus hauled is comparatively small. Wood pit-cars of three-ton capacity are used and are brought to the surface in trips of forty-five cars.

Coal from the No. 6 level is brought to the main parting by a 75 h.p. electric hoist, located on the main gangway at the head of the slope.

The No. 6 level district is ventilated by a four-foot Jeffrey exhausting fan, driven by a 30 h.p. motor. Its capacity is 50,000 cubic feet of air per minute against 1.2 in. water gauge. The outside district on the No. 5 level is ventilated by a five-foot Keith fan, driven by a 50 h.p. motor, and having a

capacity of 80,000 cubic feet per minute against 2-in. water gauge; and the inside district by a four-foot Jeffrey fan, driven by a 25 h.p. motor, and having a capacity of 50,000 cubic feet against 1.2-in. water gauge. Ventilation in the inside district on No. 3 level is similar to the last, excepting that the fan is driven by a 35 h.p. motor.

Edison electric storage battery safety lamps are used throughout the mine. Fire-bosses are equipped with Wolf safety lamps. Daily readings are taken with the Birrell gas detector.

The coal is conveyed by a rubber belt conveyor from the Head-Wrightson rotary dump to a steel tipple, which contains picking belts and shaker screens. It is separated into three sizes: over 6-in., through 6-in. and over 3-in., and through 3-in. Coal over these sizes is picked by hand. That under 3-in. is conveyed by belts to a pneumo-gravity dry cleaning plant.

This plant is the first of its kind to be installed in a coal mine in Canada, and is equipped with six air-cleaning tables. Coal is elevated to the top of the building and passed over a bank of 'hummer' electrically vibrated screens, sized as follows: 0 in. to $\frac{1}{8}$ in.; $\frac{1}{8}$ in. to $\frac{1}{4}$ in.; $\frac{1}{4}$ in. to $\frac{1}{2}$ in.; $\frac{1}{2}$ in. to $\frac{3}{4}$ in.; $\frac{3}{4}$ in. to $\frac{15}{8}$ in.; and $\frac{15}{8}$ in. to 3 in.

Coal passing through these screens goes to a hopper from which it is fed to a cleaning table. These tables or separators briefly consist of a perforated deck through which air is forced by a centrifugal fan. The top of this deck is riffled and reciprocated in the direction of the riffles by a head motion which forces the material forward. A side slope of the deck causes coal to flow over the table to the side, assisted by the air, and the reciprocation of the table causes the refuse to advance to the end of the table between the riffles. This combination of motion and air causes the delivery of refuse on one end, gradually grading to clean coal at the other end.

Each table has three discharges: the clean coal, which is fed by a belt back to the tipple and loading bins; the middlings, which are carried back to the top of the plant for re-treatment; and the refuse, which is fed to a bin for disposal at various points.

The mine office and lamp house building is of concrete, hollow tile, and stucco, the lamp house being equipped with rectifiers, motor and charging racks for the Edison storage battery lamps, two Sullivan low compressors of 1,500 cubic feet, free air at 125 pounds, and one Ingersoll-Rand compressor of 1,200 cubic feet per minute at 125 pounds, which furnishes the power for the compressed-air picks in the mine.

A commodious concrete and hollow-tile wash-house equipped with individual steel lockers and separate shower rooms, as well as a first-aid room, is furnished for the employees, hot and cold water being on tap at all times.

The Company's mines are also served by a complete foundry and machine shops, under the name of the Blairmore Iron Works. The foundry is equipped with five cupolas, brass furnace, core ovens and tumbling barrels, and complete moulding equipment. The blacksmith shop contains puncher and shearing machinery capable of shearing 1 in. by 8 in. flat-bar or punching 1½-in. holes in 1-in. stock; an air hammer which delivers 150 600-lb. blows per minute; complete forging equipment for any size job; and a complete electric and oxyacetylene welding outfit. In the machine shop are four lathes, ranging from 8-in. swing, three feet between centres, to 65-in. swing and 19 feet between centres; complete pattern and woodworking machinery; planing and threading machines; and a 200-ton hydraulic press.

Power for the operation of both Greenhill and Bellevue mines, and the Blairmore Iron Works, is purchased from the East Kootenay Power Company, whose plants are located at Bull River and Elko, B.C., some seventy-five miles west of Blairmore.

McGillivray Creek Coal & Coke Co., Ltd.:

The McGillivray Creek Coal & Coke Company, Limited, of Coleman, Alberta, commenced operations in 1909, on the S.W. quarter of Section 17, Township 8, Range 4, west of the fifth meridian.

They are operating what is known locally as the No. 2 seam, which has a dip of thirty degrees to the west, and a strike practically due north. It has an average thickness of about nine feet, with sandstone roof and shale floor. Mining

is on the room-and-pillar system, with ten-foot rooms. In the upper workings, these are turned off every sixty feet, leaving fifty-foot pillars; but in the lower workings, where the cover gets above eight hundred feet, they are at ninety-foot centres, leaving eighty-foot pillars.

Another seam, known as No. 4, has also been opened up. The condition of roof, floor, dip, etc., are much the same as in the No. 2 seam, but the coal is very much less in thickness,

averaging about five and a half feet.

The Company has been experimenting for the last two years with shaker conveyors, and recently also with long-wall coal cutting machinery, with an idea of working out some other system of extracting the coal, as the present system, whereby the coal is taken down chutes, makes a tremendous amount of coal dust (on account of the friable nature of the coal), and gives a product which contains a big percentage of slack coal.

The underground haulage is done by H. K. Porter compound air locomotives, the largest weighing ten tons, and the gathering locomotives seven tons. The coal is hauled out through a slope 3,000 feet long, driven across the pitch and having a grade of about seventeen degrees. The hoist for the slope is an electrically driven 500 h.p. machine. The mine is ventilated by two fans, one a Sirocco, with a capacity of 200,000 cubic feet per minute, and the other a Jeffrey, with half that capacity. The pumping is done by two four-stage electrically driven Ress turbo pumps, each driven by a 200 h.p. direct connected motor.

The tipple and air-cleaning plant is situated about one mile from the mine, and the haulage to the tipple is by trolley locomotives, through a 4,000-foot rock tunnel and about a 1,500 feet outside haul. The tipple is equipped with one power rotary dump. One 250 tons per hour Marcus screen takes out all coal under three inches, which is taken to the air-cleaner for treatment. The balance of the coal is carried by a 36-in. belt conveyor to a loading bin. The air-cleaning plant consists of an adequate number of hummer screens and pneumatic separators with a capacity of about 150 tons per hour. The loading equipment consists of one Christy loader, and one Ottumway loader, both electrically driven; one

generator driven by a Curtis steam turbine, and one driven by a Goldie-McCulloch Corliss-valve reciprocating engine; and two low pressure compressors and one four-stage high-pressure compressor.

At the present time practically all the electric power is supplied by the East Kootenay Power Co., and the connected

load is about 3,000 h.p.

Hillcrest Collieries, Limited:

Situated a little over one mile southwest of Hillcrest station on the Crow's Nest Pass branch of the Canadian Pacific railway, the Hillcrest Collieries, Limited, operates in a basin of coal which is cut off from the main basin by a fault running through Frank. The coal seams occur in the Lower Cretaceous immediately overlying the Jurassic.

The basin has a northwesterly trend and averages some 6,600 feet in width. On its eastern side, the measures dip

30°W., and on the western side 85°E.

Development is carried out by two parallel slopes 600 feet apart, situated on the eastern outcrop, and driven across the pitch at an inclination of about twenty-four degrees. Levels are then broken off five hundred to six hundred feet apart, those to the north circling the basin and developing the western limb, those to the southeast following in a straight line the eastern boundary of the property. The coal varies in thickness from a few feet to forty feet, the general average being about fourteen feet.

In the past, the method of extraction has been by driving rooms up the pitch from the various levels and allowing the coal to gravitate down to mine cars placed to receive it at the room mouths on the levels. The method now being adopted is to extract the coal by rooms across the pitch. All haulage on the entries is by horse, delivering to the main slopes, from which the coal is raised to the surface by main rope haulage operated by steam.

The No. 1 slope has been driven down five thousand feet,

the cover at this point reaching eighteen hundred feet.

The mines are situated about 2,500 feet from the rotary dump, to which the mine cars (each containing 4,000 lb. of coal) are hauled by steam locomotive, the dump discharging

on to a Marcus picking table having a capacity of about three hundred tons per hour. This, in turn, discharges to a retarding conveyor capable of handling the same quantity and situated 252 feet above the loading tracks. The conveyor is of the flight type and 650 feet in length, the inclination being thirty-three degrees. It delivers into bins having a capacity of three hundred tons, from which the coal is loaded into box cars by a Christy box-car loader. The tipple is connected with the main line of the Canadian Pacific railway by a standard gauge spur about one and a half miles in length, which is owned and operated by the Company.

Two other coal seams are met with in this horizon, the first, ten feet in thickness, being about 75 feet below the seam at present being worked, and the other about 150 feet deeper

and eight feet in thickness.

To the east of this basin, and separated from it by an anticline of considerable size, there is another basin corresponding to that lately operated by the Franco-Canadian Collieries, Limited. Four miles of this basin, extending in a southeasterly direction from Hillcrest station, is owned by Hillcrest Collieries, Limited. The basin averages at this point 4,000 feet in width, and a drill hole put down a few years ago at Hillcrest station shows a seam of coal of good quality and twelve feet in thickness.

The Company generates its own electrical energy, most of which is utilized for pumping purposes, the water being exceptionally heavy at certain periods of the year.

The International Coal & Coke Company, Limited:

The International Coal & Coke Company, Limited, acquired the property now being operated at Coleman in the early part of the year 1903. The property consists of about twenty-four quarter sections, roughly $1\frac{1}{2}$ miles wide by $6\frac{1}{2}$ miles long in a northerly and southerly direction.

While there are five seams on the property, only two, namely, Nos. 2 and 4, are being worked, and these are respectively fourteen feet and seven feet thick. During the summer of 1903, prospecting work was done on these two seams, and had been pushed so far that, in conjunction with surveys and prospect work done over the outcrop, both the quality and extent of the seams had been demonstrated. To be able to determine first-hand the coking qualities of the coals, a pair

of bee-hive ovens were erected at the mine and tests made, these tests showing that both seams, and more particularly No. 4, were of coking quality.

The Company being thus assured of the possibilities of the property, serious preparations for its development were started. The townsite was surveyed and put on the market, plant and plant buildings were designed, and materials ordered, and by October, 1904, the mines were equipped for and producing a large tonnage.

From that time to the present, development of the property has been systematically carried on and kept well in advance of extraction. At the end of 1926, No. 2 and No. 4 entries, driven at tipple level, have been advanced 14,550 feet and 12,300 feet respectively. A slope driven in No. 4 seam has been sunk to a total depth of 2,500 feet. At the various levels, Nos. 4 and 2 seams have been connected by short rock tunnels below the 'A' or tipple levels, and three other levels have been turned off, namely, 'B', 'C' and 'D' levels, on both Nos. 2 and 4 seams. The combined lengths of all these levels, including the two 'A' levels, is approximately 11 miles.

North of the Canadian Pacific railway, development of a parcel of property comprising about 140 acres was commenced in 1918. To date, this has been almost completely developed, and pillar extraction has been carried well forward. At a proper distance below river level, entries have been driven connecting this mine with the property on the south side.

Method of Working.—The method of working used throughout the Denison colliery is the double entry, and recently the triple entry, room-and-pillar panel system. Two or three entries are driven on the strike of the seam, with a block sixty or one hundred feet between, depending on the depth of the levels being driven. Cross-cuts between entry and counter are driven every sixty or one hundred and twenty feet, depending also on the amount of overlying strata. From the upper, or counter, rooms ten feet wide are driven straight up the pitch, and have a sheet-iron chute built on one side, and a travelling way on the other. For the purpose of ventilation, cross-cuts are driven on the strike from room to room, leaving the coal blocked out into pillars. Owing to the fact that the overburden increases as the workings advance to the

south, no standard size of block is maintained; the size increases proportionately with the cover. The smallest pillar adopted has been fifty feet by sixty feet, and to date the largest are 112 feet by 120 feet. The former are obtained by driving a room off the counter at every entry cross-cut, and connecting these rooms by a cross-cut every sixty feet up the pitch. In the latter, rooms are driven off the counter at alternate cross-cuts, and room cross-cuts every 120 feet. Every fifteen rooms, a block of coal 120 feet wide without break-through, and running from counter to entry above, is left, thereby dividing the workings into panels, tending to localize trouble from caves, squeeze, creep, etc.

Description of Plant.—Since the inception of the mine, the motive power used for haulage, pumping, drilling, etc., below ground has been, generally, compressed air, and for tipple, shops, etc., above ground, electricity. To the units first installed have been added items as required, until in 1926 the plant consisted of the following:

- 3 4-stage high-pressure compressors, 1,000 lb. pressure.
- 5 7 in. by 14 in. compound locomotives.
- 1 7 in. by 14 in. simple locomotive
- 1 7 in. by 12 in. "
- 4 6 in. by 10 in. "
- 2 5 in. by 10 in. "

Five-inch air lines, fitted with drain valves at necessary points, carry air into the mine, on the entries. Pipe is reduced to 4-in., 3-in., and 2-in., as it advances to the limit of the haul.

One 'American Well Works' electro-turbine pump, 2,200 volts, 1,000 Imperial gallons per minute, pumps water from the main pump at 'B' level to surface, and a similar but smaller unit pumps from 'C' level to 'B' level. General air-driven pumps are used in various parts of the mine, supplied with power from an electrically driven low-pressure compressor in the power house.

Hoisting over the main slope, South mine, is done with one 700 h.p., 2,200-volt, Harland electric hoist. A further slope is driven across the pitch, equipped with a steam hoist, 12 ft. by 16 ft., which will be used for lowering materials, and,

if required, for hoisting coal. At No. 8 North mine, north of the C. P. Ry. tracks, hoisting is done by a 220 h.p. Vulcan electric hoist.

Three fans are used to ventilate the workings, one at the North mine and two at the South mine. The North mine fan is an improved Murphy, belt-driven from a 150 h.p. Westinghouse motor; speed of motor 600 r.p.m., speed of fan 210 r.p.m. It has a capacity of 150,000 cubic feet per minute, and is reversible.

The South mine workings are ventilated by a Sirocco fan, belt driven from a 200 h.p. Westinghouse motor, and with a capacity of 300,000 cubic feet per minute; and a Capell fan, belt driven from a 150 h.p. General Electric motor—speed of motor, 600 r.p.m., and of fan, 243 r.p.m.—and having a capacity of 150,000 cubic feet per minute.

Any small fans used for ventilating rooms driven ahead, or rock drills in operations below ground, are driven by low-pressure compressed air.

Surface Plant.—After being in continuous operation since 1903, during which time fully six million tons of coal have passed over it, the old tipple and screening apparatus at Coleman is to be dismantled, and this season the entire output of the colliery will be handled through a new modern plant which is now ready for operation. The coal will be cleaned by the pneumatic or dry-cleaning process, which is the latest development in coal preparation, there being, so far, only two installations of this type in Canada—both, by the way, in Alberta—and very few of them on the continent.

It is predicted, however, that, as time goes on, more Western mines will adopt this principle as being best suited to our climate, which, during the busy winter season, and under the wet-wash process, creates much difficulty by the coal freezing in solid masses before it has had time to dry after passing through the washery. Elimination of this problem, as well as the improved facilities for sizing and grading the coal, opens up great possibilities for mines equipped with this type of cleaning apparatus.

The new modern tipple and cleaning plant is of steel and concrete construction and embodies the latest principles of operation and maintenance. It is the result of much close study of the relative merits of varied types of coal handling apparatus, and was adopted as being the most efficient and suitable of its kind from every angle, including sizing, cleaning, capacity, facilities for handling and loading with the minimum of breakage, and, finally, grading to suit all requirements of the steam and domestic coal markets.

Actual construction had to be carefully planned to be independent of the existing plant, though making it possible to connect up with the loading and mine tracks without delay. In operation, the plant is, as far as humanly possible, automatic. Briefly described, this system of handling coal begins at the tracks leading from the mines, where the cars are fed in a continuously coupled trip through a revolving dump by an air-driven ram, each car being weighed just before it reaches the dump.

The empties, still coupled together, then pass slowly back to the mine, the coal being fed to an inclined belt-conveyor which carries it to a Marcus screen on the tipple floor, where it is separated into two sizes—above and below three inches. On this screen the larger size is hand-picked, going directly to the cars, or is re-mixed with the cleaned coal from the washery as desired. The size below three inches is elevated to the top of the building whence it passes over hummer electric vibrating screens, which size it to suit the six separating or cleaning tables to which the product is immediately conveyed by The hummer vibrating screen is simple in various chutes. construction but highly efficient, and consists of suitably sized wire mesh stretched tight over a steel frame. To the mesh are imparted vibrations at high frequency, which transmit gentle agitation to the coal, with resulting thorough separation, the vibrations at the same time preventing clogging of the mesh.

After passing over, and being separated by, the screens, the coal is led through chutes to the tables, the flow being controlled by an automatic feeder at the head of each table. Although these tables have only recently come into use for cleaning coal, it may be noted that the same principle is widely used for grading nuts, grain, etc., and it was its marked success

in this direction that led to its being applied to coal preparation. International is the second Canadian colliery to use this method, the first plant having been installed by West Canadian Collieries at Blairmore some time ago.

The mechanical features of the plant are simple, air being used as the floating medium instead of water. The coal to be cleaned is first properly sized, the quality of the finished product largely depending upon this operation.

The separation by this process is secured by taking advantage of the difference in weight between a piece of coal and a similar sized piece of the refuse or rock to be removed.

An ordinary blower-fan is fitted to a suitable frame, connected to an air chest, and placed in position under the The top of the air chest is the deck of the table, and this deck is either perforated plate or close wire mesh, and is the separator proper of the machine. The deck, by a suitable drive, is given motions that will propel the coal from one end to the other in short strokes. The table at the same time is sloping in the direction of travel and also to one side. The air from the fan is forced through the perforations in the deck, forming hundreds of tiny air jets at suitable pressures, this pressure tending to float the coal. On the deck are a number of riffles, or partitions, higher at the intake than the discharge end. As the load is travelling over the deck, and being floated by the air, the heavy particles tend to settle to the bottom and remain within the partitions, but the lighter pieces, getting to the top, pass over the riffles, gradually travelling sideways. An intermediate product between heavy and light—the bone n the coal-does not get a chance to be lifted and to climb over the riffles until these become lower nearer the discharge end. It will be seen that the particles having different specific gravities not only stratify themselves in layers at top, middle, and bottom of the deck load, but also take different paths and eventually discharge each at a given point. When the products finally reach the end of the table, they are discharged into separate chutes which deliver on to travelling conveyors whose duty it is to carry the coal to storage bins ready for shipment. The tipple screens, hummer screens, and air tables are designed to separate and clean coal to the following sizes: 3 in. and over; 3 in. to $1\frac{5}{8}$ in.; $1\frac{5}{8}$ in. to $3\frac{3}{4}$ in.; $3\frac{3}{4}$ in. to $3\frac{3}{8}$ in. to 0 in.; thus giving a range of sizes to suit all requirements.

While the use of air at pressure would ordinarily tend to create dust within the building, this plant has in conjunction with its machinery an elaborate system of dust collection. The tables have closely fitted hoods, which are connected by pipes to two large fans and collectors, the function of which is to draw off the dust as made and deposit it in the collectors. The screens are entirely enclosed, thus leading any dust made directly to the tables, where it is drawn off as above described.

With these most up-to-date facilities for handling, screening, sizing, and cleaning the coal, the International Coal and Coke Co., Ltd., is now in a position to ship any size of coal required by their customers, free from impurities and low in ash, and will be able to compete with the best United States coals which have previously maintained such a strong foothold in the Manitoba market, and at times even further west.

The machine shop is a fireproof concrete and steel building, fully equipped for doing all classes of mine repairs and building, and includes power hammer, lathes, planer, drill press, saws, acetylene welding outfit, etc. The Company build their own cars, buying the irons and wheels made up.

The locomotive shed also is a fireproof concrete and steel building, and has ample pit capacity for repairs and storage of locomotives. It is steam-heated and equipped with steam jets for cleaning locomotives.

The power-house equipment comprises the three high-pressure compressors, and the low-pressure compressor previously mentioned. The latter is now being replaced by an electrically driven Ingersoll-Rand machine of 2,375 cubic feet capacity. One mixed pressure Bellis and Morcom turbogenerator is maintained as a standby source of power, but power is purchased from the East Kootenay Power Company. As the high-pressure compressors are replaced, electrically driven units will be installed, as the Company is working towards complete electrification.

The boiler house installation comprises six Leonard & Sons return tubular boilers, working at 150 lb. pressure.

Besides the above there is a commodious bath-house for workmen, fitted with individual steel lockers, and also hot and cold water showers; a mine office for officials; warehouse; oil house; electricians' workshop; sawmill, sand-house and drier, and various other smaller buildings necessary to efficient operation.

Coal Estimates, Production and Reserves.—Based on Nos. 2 and 4 seams only, with a total thickness of 21 feet, the estimated quantity of coal on the property is 108,370,000 tons.

The following table gives the amount of coal and coke produced since 1904, and their value:

Year	Coal		Coke			
	Tons	Value	Tons	Value		
1903-4	50,120	\$	2,191	\$ 8,791		
1905	173,023	293,475	13,306	50,639		
1906	334,230	553,553	31,067	31,067 114,429 39,122 148,691 50,491 194,845 51,070 198,338 69,645 237,700 22,116 84,846 81,910 318,927		
1907	372,480	675,184	39,122			
1908	445,180	823,925	50,491			
1909	370,527	679,442	51,070			
1910	475,080	876,762	69,645			
1911	138,906	254,903	22,116			
1912	425,131	774,693	81,910			
1913	413,646	773,352	65,103	254,366		
1914	239,527	458,782	28,542	111,751		
1915	106,962	191,642	24,187	92,807 179,356 182,000 216,994		
1916	209,572	419,244	42,548			
1917	187,992	508,086	31,175			
1918	252,858	829,780	32,801			
1919	149,974	347,936	Not operating			
1920	324,870	1,415,539	•			
1921	147,184	708,725	46			
1922	113,369	518,256				
1923	253,857	1,120,394	66			
1924	116,090	500,243	66			
1925	178,542	669,706	66			
1926	233,946	788,377	"			
Total	5,713,066	\$14,282,499	585,274	\$2,424,480		

From the above table, it is seen that the total production of coal to the end of 1926 is 5,713,000 tons, leaving reserves still to be mined about 102,657,000 tons, which assures operation, even at present maximum output, for many years.

Outside of what coke has been used by the smelters, the Canadian Pacific railway has provided the main market for the Company's output.

Central Power Plant.—Individually, the mines are all well equipped with steam power plants, but provision has recently been made for connecting the mines with hydro-electric energy.

The East Kootenay Power Company are successors to the British Columbia and Alberta Power Company, which was formed some two or three years ago by Minneapolis interests who proceeded to develop a plant at the falls on the Bull river. The installation consisted of two 2,500 k.v.a. 2,300-volt, 3-phase, 60-cycle, (maximum rated) generators, driven by vertical water-wheels. The equipment included direct connected exciters and the necessary auxiliaries, with switchboard and voltage regulator.

The capacity of this plant was found to be too small to meet the full demand for power in the district served, which extended from Kimberley and Cranbrook on the west, to Bellevue on the east.

The serving of this district required a transmission line of approximately one hundred and twenty miles in length with step-up transformers at Bull River station and a substation for each customer to be served. An aluminium line was run from Bull River to Cranbrook, and also from Bull River to Bellevue. Later a line was run from Bull River to Kimberley, and it is proposed to sell a large block of power to a concentrator being built at that point. The line voltage is 66,000.

The Company used wood-pole line construction with aluminium cable throughout. On the eastern portion of the line, that is, east of The Divide, H-frame or two-pole construction was used. This was necessary on account of the heavy winds encountered in the Crowsnest Pass district in Alberta.

Seeing that the Bull River plant would be quickly overloaded, the Company decided to develop power at Elko, a short distance below the Elk falls. This plant consists of two 2,600 k.v.a., 6,600-volt, 3-phase, 60-cycle, water-wheel driven units, and provision will be made for the later addition of a third unit of the same size. Each unit is equipped with direct connected exciter and separate excitation is provided for in addition.

At the Elko plant, the Company have installed the very latest equipment in respect to generating and control equipment. The control equipment is designed to take care of load fluctuations and to provide for the minimum possible amount of interruptions, with a view to giving a better service to the customers in the Crowsnest Pass district.

On account of adverse comments regarding the operation of an electric system of 120-mile single line, the Company are installing a steam stand-by plant at Sentinel, B.C. This plant will be equipped with a 5,000 kilowatt steam turbine and powdered fuel will be used under the boilers. This will ensure to customers in the Crowsnest Pass district a reliable source of energy. It will now be quite safe for coal companies to instal electric motors for the driving of their compressors and other similar machinery, and also of their mine fan.

The Company have arranged for modern lightning-arrester protection for both plants, this being very necessary in this district. Thus, excellent protection is provided against

interruptions from this cause.

Comments have been made with respect to voltage conditions at the end of the transmission line, but the Company have signified their intention of providing, at the earliest possible moment, corrective equipment to take care of any irregularities in the voltage. This corrective equipment will work automatically as the load increases and, therefore, it is reasonable to expect that in the course of a few months the voltage supplied will be satisfactory to all.

At the present time, the customers being served are:

The Consolidated Mining and Smelting Company of Canada, at Kimberley, B.C.

The Corporation of Cranbrook, B.C.

The Corporation of Fernie, B.C.

The Crow's Nest Pass Coal Co., Ltd., Coal Creek, B.C. The McGillivray Creek Coal and Coke Co., Ltd., Coleman, Alta.

The International Coal and Coke Co., Ltd., Coleman, Alta. West Canadian Collieries, Ltd., Blairmore and Bellevue, Alta.

Customers installing electric power for their plants should make sure that they put in the necessary protective and control equipment. Circuit breakers must be made large enough to rupture any short circuit that it is possible to build up under the particular conditions involved, otherwise serious trouble may result for the user. Protective equipment of ample capacity is so easily procurable that it is quite unnecessary to take any chances.

The policy of the East Kootenay Power Company is, evidently, to give their customers the best service possible, under the conditions to be met with in this district. The market for power in the district served is excellent, and as all of the customers possess marked financial standing, the business should be profitable. The power service should be equally profitable to the customers. By the use of electric power they will be enabled to dispense with large steam plants, entailing heavy operation and repair expense, thereby avoiding the costly item of stand-by losses.

The Canmore-Bankhead Field

While the coal of the Kootenay formation is chiefly bituminous, in the Canmore-Bankhead district it has been altered to semi-anthracite and anthracite, the greatest alteration being found near Banff.

The Kootenay coal measures are here underlain by Fernie shales (Jurassic) resting on Carboniferous limestone, and are overlain by shales and sandstones of the Dakota series. They are intensely folded and faulted as a result of the orogenic movements which accompanied the upbuilding of the Rocky mountains, and occur in a basin, or trough, between parallel ranges of limestone mountains—to which circumstance the coal measures owe their escape from erosion. Some idea of the magnitude of the forces to which the region has been

subjected may be gained from the fact that, although the Kootenay coal measures are Lower Cretaceous and stratigraphically very much below the Belly River and Edmonton groups, they are found here at a much greater elevation than these latter, as they occur at Lethbridge and Drumheller respectively; while the Carboniferous limestone in the immediate vicinity has been lifted above the Kootenay measures and in places thrust over them.

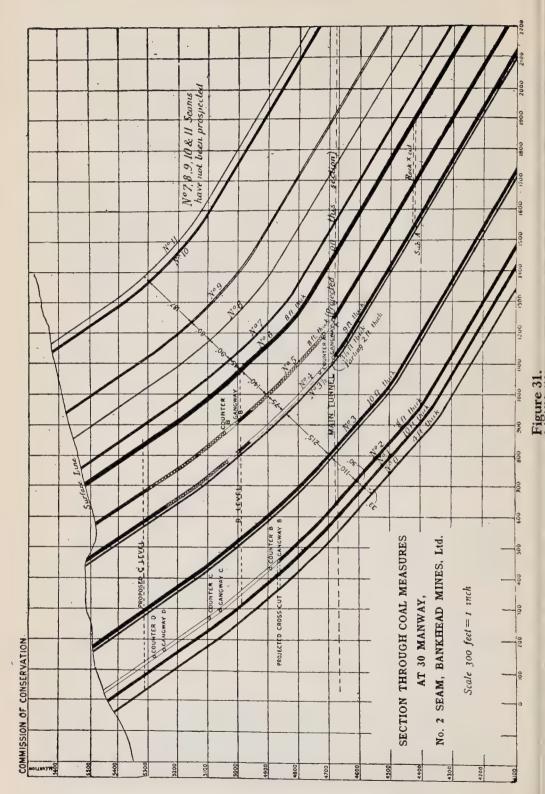
The basin has a general N.20°W. strike, and extends along the Cascade, Bow, and Kenanaskis rivers in the Rocky Mountain park, on the main line of the Canadian Pacific railway west of Calgary. As is to be expected, the dip of the measures is extremely variable. Over most of the field it ranges from 40° to 70°, but in some sections it is as low as 20°, and in a few places even less than this.

At the northern end of the basin, the coal is an anthracite, and at the southern end a low-volatile steam coal. The former has been mined extensively at Bankhead, and also at Anthracite; the latter at Canmore and Georgetown. The mines at Canmore are the only ones now operating in the district.

The following are representative analyses of the two classes of coal:

	1		
	Canmore	Bankhead	
Moisture	1.00	1.00	
Volatile matter	15.00	9.00	
Fixed carbon	74.00	78.00	
Ash	10.00	12.00	
	100.00	100.00	
Sulphur	0.80	0.60	
B.t.u	13,500	13,500	

With the building of the Canadian Pacific railway through this district in 1883, and its operation in 1884 as far as Canmore, active development of the field commenced. The first opening was made in 1886 by the Canadian Anthracite Coal Company



at their No. 1 mine, to which a branch line was built in 1891. This was the only steam coal mined in the Province until the opening of the Frank mine in 1901.

Thomas Cochrane and others opened a mine to the north of this, on School Section 29, in 1888, but on account of the faulted condition of the seams, this mine was closed down in 1893 and the property acquired by the H. W. McNeill Company, who afterwards turned it over to the Canadian Anthracite Coal Company.

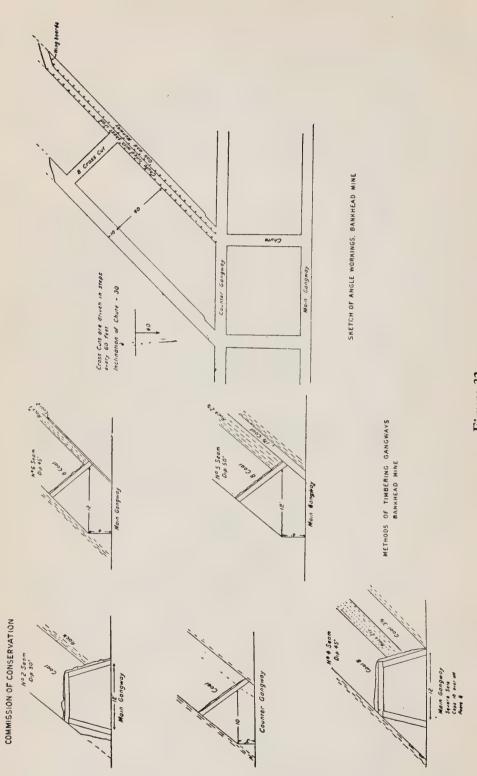
Anthracite Mines:

Anthracite Mine.—This mine was opened in 1888 by the Canadian Anthracite Coal Company, and worked until 1904, when, the property having been exhausted, it was abandoned. Five seams were worked, varying from 3 ft. to 8 ft. in thickness, and designated, from the bottom up, the Nos. 1, 2, and 3, and the 'A' and 'B' seams. The mine was opened through a slope below water level, at an elevation of 4,600 feet, and sunk one lift on the No. 2 seam, with rock tunnels across the measures to the other seams worked, and an underground slope to the 'A' seam.

Bankhead Mines.—This mine, operated by the Department of Natural Resources of the Canadian Pacific Railway Company, was opened in 1903-4, and the plant was completed and put into operation in November, 1905. Several seams have been worked, as stated in the table below. They are

Seam No.	Thickness	True thickness of intervening strata	Remarks
0	4 ft. 0 in.	33 ft. to No. 1 seam	Workable, but not worked
1	10 ft. 0 in.	20 ft. "No. 2"	46
2	8 ft. 0 in.	110 ft. " No. 3½ "	Worked
3	10 ft. 0 in.	215 ft. " No. 3½ "	4.6
4	12 ft. 6 in.	75 ft. " No. 5 "	6 6
5	8 ft. 0 in.	140 ft. " No. 6 "	4.6
6	8 ft. 0 in.	45 ft. " No. 7 "	4.4
7		90 ft. " No. 8 "	Prospected
8		80 ft. " No. 9 "	4.6
9		187 ft. " No. 10 "	4.6
10		19 ft. " No. 11 "	46





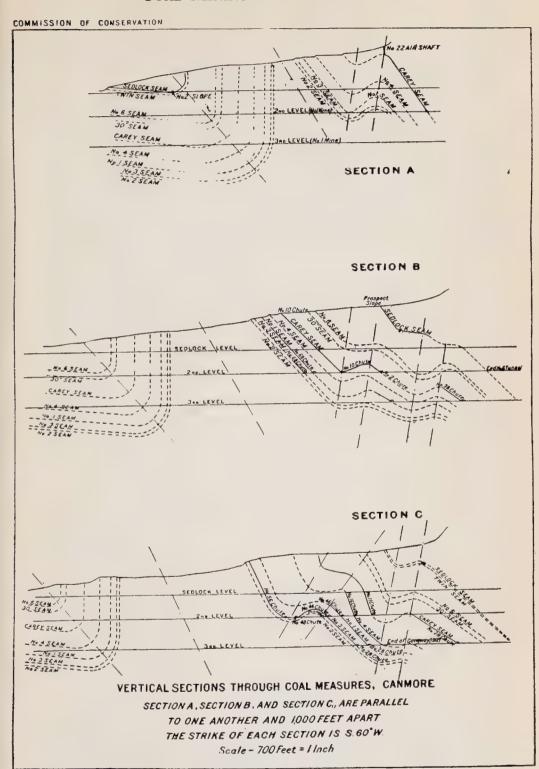


Figure 33.

numbered, from the bottom up, No. 0 to No. 10, and they vary in thickness from 2 ft. 6 in. to 12 ft. 6 in., with local enlargements and corresponding contractions.

The Bankhead mine was opened by drifts at three levels, all above water level, at elevations respectively 4,650, 4,980, and 5,234 feet above sea level, with rock tunnels across the measures to the several seams, and the coal was lowered on outside inclines. An inside slope was later sunk from the lowest level to an elevation of 4,300 feet, with gangways on three seams. This filled up with water during the long strike of 1911, and it has not been pumped out. The Bankhead mine was closed on March 31st, 1922.

A feature of the operations was the briquetting plant, utilizing the dust resulting from the mining and sizing of the coal. The first unit was started in March, 1907, and the second added in 1908, giving the plant a capacity of 600 tons of briquettes per day when both units were run continuously.

Steam-coal Mines:

Canmore Coal Company, Ltd.—This mine is situated on the right bank of Bow river, west of and adjoining the town of Canmore. A short spur connects the mine with the Canadian Pacific railway. In all, eight seams have been worked on this property, designated, from the bottom up, Nos. 1, 2, 3, 4, 5, and Carey, Sedley, and Stewart seams. The mines were worked under lease by the H. W. McNeill Company from 1891 to 1911, since which time they have been operated by the Canmore Coal Company.

No. 1 mine was opened by a slope sunk two lifts below water level and an underground slope for a third lift, with rock tunnels across the measures to the other seams worked; the elevation at the mouth of the slope being 4,420 feet, and at the second and third lifts 4,072 feet and 3,865 feet respectively. This mine was closed in 1914.

No. 2 mine was opened with a slope below water level, at an elevation of 4,368 feet, sunk one lift to 4,307 feet, with rock tunnels across the measures to the other seams; and by underground slopes to elevations of 4,215 feet, 4,123 feet, 4,046 feet, and 3,995 feet, respectively, with rock tunnels across the

measures to the other seams worked. This mine was opened in 1907 and is still in operation. Two seams, each five feet in thickness, are being worked, and, owing to the measures being sharply folded, the dip is very variable, from horizontal to vertical. The mine is developed by slopes, and levels are driven to give lifts of about 800 feet. The method of mining is room-and-pillar on the flat or low-dip portions of the seams, and the angle system where the dip is steep or vertical. Compressed-air locomotives are used for haulage underground.

Canmore coal is a high-grade semi-anthracite and is used principally as a locomotive fuel, although a portion of the output is used for heating and power purposes. In the screened sizes it makes a good grade domestic fuel. About 99 per cent of the coal is shipped as mine run. The Company has recently installed a briquetting plant, which will utilize the small coal. These briquettes make a high-grade domestic and steam fuel.

Georgetown Mine.—This mine was opened in 1913 by the Canmore Navigation and Coal Company, afterwards the Georgetown Collieries, Limited. In 1916 it was acquired by the Canmore Coal Company and closed up. The seam worked has a thickness of 6 feet, and the mine was opened by a slope sunk one lift below water level.

Coal mining operations in the Canmore-Bankhead field have been attended with many difficulties on account of the disturbed condition of the strata, high pitch of the seams, great thickness of overlying strata, the crushed state of the coal, and the presence of explosive gases; although the problems perhaps have not been greater than those met with in other mountainous fields.

Production to date from the district has amounted to approximately ten million tons. A portion of this has been used in making briquettes, of which the output has been some 1.300,000 tons.

Brazeau District

This coal field extends in a north and south direction for forty-six miles between the North Saskatchewan and main Brazeau rivers. The width of the coal basin has been estimated to be about seven miles.

The Brazeau mines, owned and operated by the Brazeau Collieries, Limited, at Nordegg, were first opened in 1913 to supply the Canadian Northern (now Canadian National) railways with steam coal.

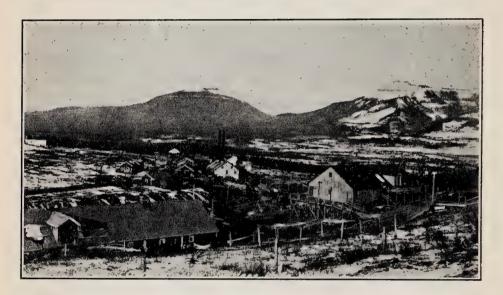
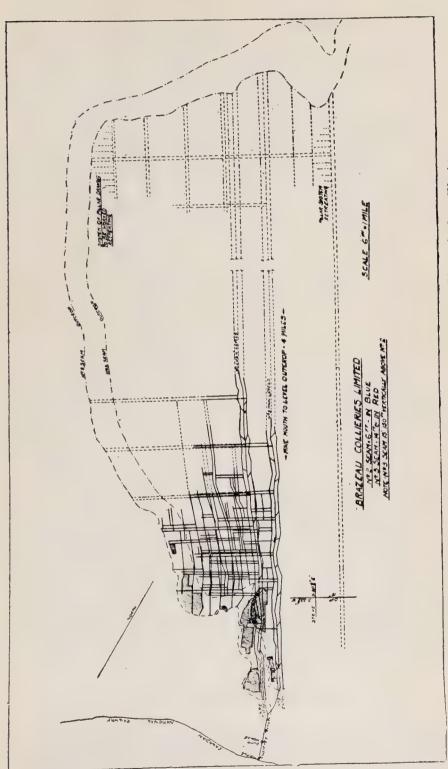


Figure 34.—Brazeau Collieries, Limited, Nordegg.

At present two seams are being developed, one a six-foot and the other a fourteen-foot seam. The mines are worked from the outcrop by tunnels driven into the seam on a 0.6 per cent grade in favour of the loads. Main tunnels are now in a distance of approximately two miles from the outcrop and they have a twelve-degree-rise lift of coal above drainage of more than 4,000 feet. It is estimated that at the present time there is developed and in pillars a reserve of over 6,000,000 tons of coal. To date the coal seams have been operated on the pillar-and-stall system, rooms being worked up the pitch and cars taken to the face by gravity balances. The coal is gathered on the upper levels and lowered to the main haulage by gravity interline drums. On the main haulage a fleet of five storage-battery locomotives is used to bring the trips of coal to the surface.

From the mouth of the main entries a ten-ton trolley locomotive hauls the coal to the tipple and cleaning plant, a distance of approximately 1,500 feet. At this point the



Brazeau Collieries Limited, Nordegg, Alberta. General Plan of Mine Workings.

Company have recently installed a new air cleaning plant of the most modern type. It consists of Arms patent screens, which size the coal preparatory to the cleaning process on Arms concentrators. The plant is equipped with exhaust collecting systems which suck the dust from the concentrators and discharge it into collectors from where it is taken to the colliery boilers. The clean coal is conveyed to a Christy box-car loader for loading into railway cars. The present tipple and cleaner have a capacity of 2,000 tons per day. To date 3,720,000 tons of coal has been shipped. The present average analysis of the coal as shipped is as follows:

Moisture 0.50 per co	ent
Vol. comb. matter	
Fixed carbon	
Ash12.80 "	
Sulphur	
B.t.u13,355	

The power generating equipment for the mine and townsite includes four 150 h.p. boilers, and the Company have under consideration the installation of a 300 h.p. rating water-tube boiler fired by pulverized coal. The boiler plant supplies steam for a 500 k.v.a. steam-driven turbine unit, and a complete stand-by unit is kept in reserve in case of breakdowns.

The townsite of Nordegg is located approximately one-half mile from the mine plant and it is claimed that it is one of the cleanest and most up-to-date mining towns in the West. In the town are four- and five-roomed cottages, bank, general store, butcher shop, bakery, community hall, and moving-picture house, hospital with fully equipped operating room, and a miners' club. The town is electrically lighted and all five-roomed cottages and main buildings are provided with running water, sewers, and baths.

Mountain Park District

This district embraces the areas being worked at Cadomin, Luscar, and Mountain Park. The mines are served by the Mountain Park Coal Branch, some 31 miles in length, which connects with the Canadian National railway at Coalspur. This branch railway, however, is operated by the Canadian National Railways System. The Luscar Collieries have constructed, from Leyland, a branch railway some seven miles in length to their collieries, this line also being operated by the Canadian National railways.

Geology.—The Mountain Park Coal Company's mines are located on the south end of the field, and are separated from the Cadomin-Luscar area by the Nikannassin limestone range. The attitude of the limestone is a large overturned fold, with the fold towards the northwest, which causes the coal measures to be repeated on the Cadomin side of the range. The coal seams dip in a northerly direction from Cadomin, and although the geology has not been thoroughly worked out it is believed that the lower measures, as well as upper measures not represented at Cadomin, occur at Luscar further to the northwest. At Cadomin, reverse dips towards the limestone are the rule, due to the fact that the area is near the base of the overturned fold. At Mountain Park the measures are more or less disturbed by rolls, but at Cadomin they are very regular.

The coal produced from this field is a very high-grade bituminous coking coal, especially adapted for locomotive use and power purposes.

Mountain Park Collieries, Limited:

This colliery is situated at Mountain Park, where operations were commenced in 1913. Development was at first by adit levels into the outcrop of the coal seams and later by sinking slopes. Four seams have been opened up but only two are now being mined, one 30 ft. thick and one $4\frac{1}{2}$ ft. thick. They have very variable dip, averaging between 15 and 40 degrees. Mining is by the room-and-pillar method, with main rope haulage underground, and main and tail rope haulage on surface. The surface plant consists of:

On tipple:

- 1 picking belt and motor
- 1 Ottumwa box-car loader.

In power house:

- 3 72 in. by 18 ft. 150 h.p. H.R.T. boilers
- 4 75 h.p. H.R.T. boilers
- 2 200 h.p. Robb-Armstrong automatic cut-off engines

1 150 k.w., 250-V., D.C. generator

1 125 k.w., 250-V., D.C. generator

1 air compressor, single stage, 400 cu. ft. per min.

2 boiler feed pumps

1 200 h.p. feed water heater

In workshops:

1 16-in. engine lathe

1 24-in. Bertram drill press

1 Racine power hack-saw

1 planer, 3 ft. by 3 ft.

On fan shaft:

1 Sheldon mine fan, 200,000 cu. ft. per min. (East mine)

1 150 h.p. fan engine, steam driven

1 mine fan, 30,000 cu. ft. per min. (West mine)

2 fan motors, 20 h.p., D.C., 250 volts

In mines:

2 electric hoists, 60 h.p., D.C., 250 volts

1 electric hoist, 50 h.p., D.C., 250 volts

2 centrifugal pumps, 35 h.p., D.C., 250 volts, 250 g.p.m. capacity

Buildings on surface:

1 hotel and annex

1 general store

1 club hall and picture theatre

100 dwelling houses

1 school

Luscar Collieries, Limited:

This colliery, situated at Luscar, commenced operations in 1922 by driving an adit level into the outcrop of the coal seam. Later development was by a slope from this level. The seam averages 35 feet in thickness, with dip varying from 20 to 50 degrees. The pillar-and-stall method of mining is employed, with main rope haulage on the slope. The surface plant includes the following equipment:

In tipple:

1 Head-Wrightson rotary dump

1 Marcus shaker screen and rock conveyor

1 Ottumwa box-car loader

All driven by electric motors and having a capacity of 200 tons per hour

In power house:

4 150 h.p., 72 in. by 18 ft. H.R.T. boilers

1 Sirocco induced draft fan

1 steam jet ash conveyor

2 5 by 7½ by 10 duplex boiler feed pumps

2 4 by 6 by 10 duplex service pumps

1 600 h.p. Cochrane feed water heater

1 187 k.v.a., 2,200-V., 3-phase, 60-cycle, A.C. generator

1 220 h.p. Bellis-Morcom compound vertical engine

1 312 k.v.a., 2,200-V., 3-phase, 60-cycle, A.C. generator

1 400 h.p. Bellis-Morcom compound vertical engine

Above units are direct connected. Panels are wired for parallel operation and include synchronoscope and voltage regulator.

1 Bellis-Morcom two-stage compound steam-driven air compressor of 1,500 cu. ft. per min. capacity.

In workshop:

1 Monarch engine lathe, 20-in.

1 Bertram drill press, 24-in.

1 Racine power hack-saw

At top of slope:

1 hoisting engine, 24 in. by 36 in. double drum, first motion

On air shaft:

1 100,000 cu. ft. per min. Sirocco mine fan.

1 100 h.p. electric motor, A.C., 550 volts

In the Mine:

2 centrifugal pumps, electric driven, 75 h.p., 370 g.p.m. capacity.

In the lamp cabin:

175 Edison electric cap lamps

100 Wolf safety lamps (not in use)

A wash-house for the workmen providing accommodation for 250 men. An office, boarding house, general store, hospital, and 90 dwelling houses. One community hall. One school.

Cadomin Coal Company, Limited:

Extending regularly across this property are three outcrops of the same seam, due to a synclinal fold followed by an eroded anticlinal fold. The thickness of the seam varies from twenty-three to forty feet and the dip from thirty-five degrees to vertical.

Owing to the character of the seams, and also to climatic conditions, the mine-fire hazard is unusually great—in fact two mines were abandoned for this cause. As a result of this experience the President of the Company, Mr. F. L. Hammond, devised a rock tunnel scheme of development, of which the following is a brief outline.

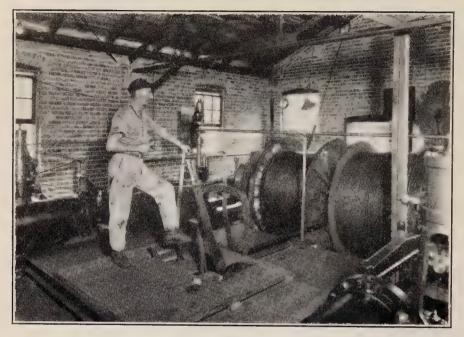
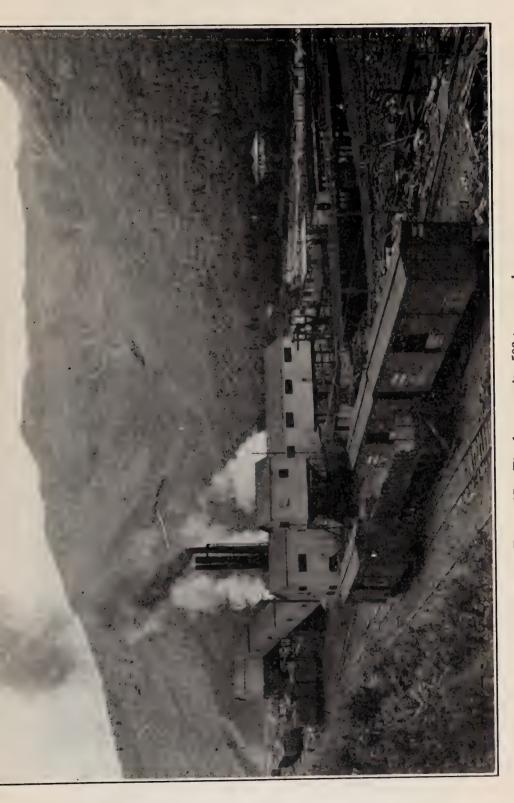


Figure 36.—Main-and-tail hoist. Cadomin Coal Company, Limited.

The mine is developed by a main and counter rock tunnel driven parallel to the strike of the seam and separated from it by fifty feet of solid sandstone. A cross-cut from each of the rock tunnels is driven to tap the coal every eight hundred feet and these are the only entrances into the panel; each coal panel is separated from the next by a coal barrier some 200 feet in thickness. The main entries in coal are therefore only about six hundred feet long and the angle system is used from the main coal entry to the surface. The average lift worked is about 1,000 feet and the coal gravitates from the working places to the main loading chutes in the main entry below. As soon as all the development in a panel is completed, pillars are drawn in retreat from the surface downward. When all the pillars are drawn, concrete stoppings are placed in the two rock cross-cuts mentioned previously, and the old gob is isolated and is therefore no longer a portion of the mine.

By this system, therefore, panels are completely isolated, and every security is given against fires and the extensive propagation of mine explosions.



Main and tail rope haulage is used in the rock tunnel. The main and tail engine is a first motion hoist, 15 in. by 36 in. cylinders, supplied by John Wood & Sons, of Wigan, England.

The boiler plant consists of two 271 h.p. Babcock & Wilcox Goldie-McCulloch boilers equipped with five-compartment forced draft chain grate stokers.

The power plant consists of:

- 1 720 h.p. Bellis & Morcom steam engine, direct connected to a 625 k.v.a. Canadian Westinghouse generator
- 1 320 h.p. Bellis & Morcom steam engine, direct connected to a 200 k.w. Canadian Westinghouse generator.
- 2 steam-driven Ingersoll-Rand air compressors
- 1 electric-driven Ingersoll-Rand air compressor having a total capacity of 2,000 cubic feet of free air per minute.
- 1 Weir turbine pump of 4,500 gallon capacity, used for feeding water to the boilers.



Figure 38.—Electrically controlled rotary car dump at tipple. Cadomin Coal Company, Limited.

Flow Sheet, Cadomin Coal Company, Limited. Tipple 30 3-ton car trips Car feeder Chain conveyor Rotary dump Coal Empty car by gravity to empty car make-up point 10-ton hopper Heavy-duty apron feeder Parrish screen, 2-in. perforations Slack Lump 42-in. belt conveyor, Picking table 17° incline 42-in. belt conveyor, Refuse 17° incline Flight conveyor 2 20-ton bins Rock dump 10-ton hopper Picked run-of-mine Manierre loader Railway cars Railway scale Slack Note: Products made... {Picked lump Market. Picked run-of-mine



Figure 39.—Oak spring-board shaking screen. Cadomin Coal Company, Limited.

Tipple.—The mine is equipped with a modern steel and concrete tipple having a capacity of 500 tons per hour. The coal is thoroughly screened on shaker screens and picked on a large steel apron picking table. Belt conveyors are used for elevating the coal to the loading house. All motors are controlled by push buttons and operated by automatic control panels.

The coal is weighed on a 150-ton Canadian Fairbanks scale of latest pattern.

The mine yard is equipped with adequate trackage facilities to handle 5,000 tons of coal per day.

Jasper Park District

This district is situated on the Canadian National railways on the Edmonton to Vancouver transcontinental line, at the entrance of the Yellowhead pass.

The general attitude of the limestone mountains at this point is an anticlinal dome. This anticline pitches northwest, and in front of Bulrush mountain, which forms the edge of



Figure 40.—Steel apron picking table. Cadomin Coal Company, Limited.

the Rockies, at Brule lake, the lowest beds exposed are of the Kootenay formation. In these beds three coal seams (Nos. 1, 2, and 3) occur, two of which are being worked by the Blue Diamond Coal Company, Limited, the only operator in this field at the present time.

Development has been chiefly in connection with the upper (No. 1) and middle (No. 2) seams, and at present work is entirely confined to the latter. The seams have the following average thickness: No. 1, 7 ft.; No. 2, 10 ft.; and No. 3, 5 feet. Owing to the bent and folded condition of the seams, the dip varies from six to eighteen degrees.

The No. 2 seam has been developed by a main slope, which, driven through rock for a distance of 1,000 feet, intersects, at an angle of 35 degrees, the deepest basin, which it has continued to follow for a total distance of 1,200 feet in coal.

A feature of the formation is the flat pitch on the right, or eastern, limbs, and a steeper pitch on the left, or western limbs. On the right limbs the pitch varies from five to forty degrees, whereas on the left limbs, in some places, it is nearly

vertical and probably averages about seventy-five degrees. Three parallel synclines have been developed, and the intervening anticlines pitch downwards more rapidly than the bottoms of the synclines, eventually forming a large flat basin, which is the development objective of the present main slope.

The main slope is started from tipple level and the scheme of development is to make a connection with the upper seam

and haul coal, eventually, through one opening.

A large quantity of coal has been mined from a high-level tunnel connected with the tipple by an inclined self-acting tramway, which has been abandoned for the time being in view of the approaching connection through the main slope.

A very large tonnage of coal is available in the upper section of the mountain, and this will be eventually opened up by tunnels and connected with the tipple by inclined tramways.

The method of hauling adopted in the main slope at present is endless rope. The method of mining is by breast

and angle system, with 100-foot pillars.

Ventilation is effected with a Walker indestructible fan having a capacity of 200,000 cubic feet of free air per minute. Each side of the mine is separately ventilated by independent splits, each landing having a separate split. Doors in the mine are completely eliminated by this system.

The new tipple has a capacity of 3,000 tons per day, and has a double rotary dump, grizzley screens, and a picking belt 100 feet long and 60 inches wide. The rock is conveyed through the rotary dump, and, together with the refuse picked from the belt, passes by conveyor direct to a bin on the railway track.

Storage capacity is available for 700 tons of coal, with three loading chutes, over the railway tracks. The accompanying chart (Figure 41) illustrates the flow of the coal from the rotary dump to the railway cars.

The power plant consists, in all, of six return tubular boilers with a total capacity of 700 h.p., a Sullivan compressor with a capacity of 2,000 cubic feet of free air per minute, and a 375 k.w. Bellis-Morcom generator. An additional unit of 400 k.w. Bellis-Morcom-engine generator of the same type is installed.

The mine is well equipped with machine shops, and carpenter and blacksmith shops, all having machine installations for doing the work necessary for the maintenance of the plant.

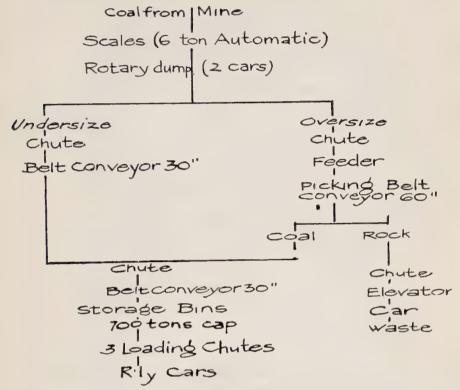


Figure 41.—Flow sheet of tipple. Blue Diamond Coal Company, Limited.

Belly River Coals

The coals that belong to the Belly River horizon have an underspread distribution. The coal is high-grade subbituminous, low in moisture and ash, and high in B.t.u's. Coal mining is being carried on in the following principal fields: Lethbridge, Taber, Saunders Creek, and along the Coalspur branch of the Canadian National railways.

Lethbridge Coal Field

The Lethbridge coal field forms a basin with its southerly outcrop at the city of Lethbridge. The axis of the basin is approximately north and south. The dips are low, but the

coal is badly displaced by a great number of upthrow and downthrow faults. There is no general direction of faulting, although north to south and east to west faults are most common. Local faults often have a displacement up to twenty-five feet. The depth of cover over the coal varies from the outcrop up to about 650 feet at the bottom of the basin. Mining has been carried on in this field for a great number of years.

Coal mining in the Lethbridge field is difficult and costly for the following reasons:

- (a) The great cover necessitates the sinking of deep shafts, and the timbering costs are high. Again, the upkeep is high due to more or less idle time in the summer.
- (b) The many faults effect the cost as well as the method of mining.
- (c) The following table indicates a more or less general section of the coal seam:

Draw slate	.0 ft.	8 in.
Coal	.4 ft.	3 in.
Bone	.0 ft.	8 in.
Coal	.0 ft.	5 in.
Clay	.0 ft.	4 in.

From this it can be seen that the total coal in the seam does not exceed 4 ft. 8 in. in thickness.

- (d) The draw slate and bone in the seam require large and expensive surface plants to efficiently clean the coal.
 - (e) During idle time the overhead is heavy.

The method of mining has been fairly well standardized in this field, after the Galt mines practice. However, market conditions during the last few years have changed somewhat, and the ideal system would provide for:

- (1) Little or no up-keep during idle times.
- (2) Concentration of operations so as to permit of large production from very little development.

A modified system of longwall or shortwall mining, if practicable, would give this result. To make a success of this system requires the adoption of a rapid method of cutting, and installation facilities for loading out a cut per shift.

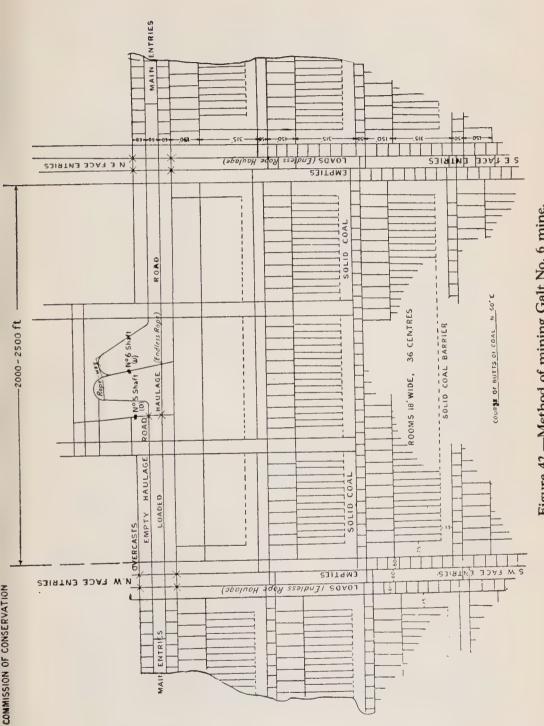


Figure 42.—Method of mining Galt No. 6 mine. Department of Natural Resources, C. P. Ry. Co.

SKETCH SHOWING ARRANGEMENT OF TRACKS ON TIPPLE

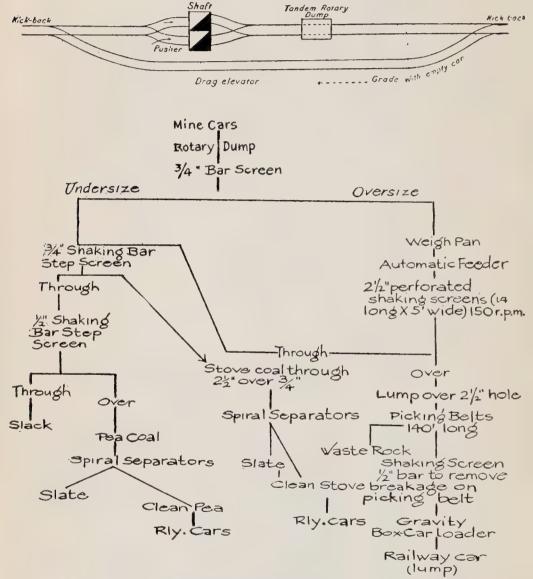


Figure 43.—Flow sheet, Galt No. 6 mine. Department of Natural Resources, C. P. Ry. Co.

Mining Methods.—All the large mines in this field have been opened up by vertical shafts, which range from three hundred feet to six hundred feet in depth.

The system of mining is the panel method, room-and-pillar, the rooms varying in width, at the different mines, from 18 feet to 33 feet, and the room pillars from ten feet to twenty feet in thickness. The accompanying sketch (Figure 42) shows the method of working Galt No. 6 mine, of the Department of Natural Resources, Canadian Pacific Railway.

The coal is undercut by compressed-air puncher-machines. The shooting is done with black powder, although the use of 'permitted' explosives is the general rule at these mines.

Endless rope haulage is used on the main roads, and horse haulage on the butt entries.

Preparation.—The coal is thoroughly sized and picked at well equipped screening plants. Screening operations are outlined in the accompanying flow sheet (Figure 43).

It will be noted that spiral separators are used for automatically separating the bone and slate from the stove and pea coals. These were introduced by the writer some three years ago, and are giving satisfaction. Unlike most coalcleaning appliances, spiral separators work on the principle of different surface frictional resistances between slate, bone, and coal, not on differences of specific gravity. This installation was the first of its kind, cleaning other than anthracite coal, to be installed in Canada.

Markets.—The coal is used principally for domestic purposes in the four western provinces, but a very small portion is shipped to the neighbouring State of Montana.

Saunders Creek District

This is a comparatively new field, situated on the Canadian National railway between Rocky Mountain House and Nordegg. Although it is believed that the coal seams in this field belong to the Belly River formation, the moisture content of the coal is about four per cent lower than in the Lethbridge field, on account of the more intense folding here. There are two known workable seams, No. 1 and No. 2, with some sixty feet



Figure 44.—Spiral separator, for automatic removal of impurities from coal; Galt No. 6 mine.

Department of Natural Resources, C. P. Ry. Co.

of measures in between. The lower, or No. 2, seam is about four and a half feet thick and its dip is about six degrees. It is not possible to give a description of each of the small mines in this district. The following is a description of the Saunders and Bighorn Coal Company's mine, which is the largest in the field.

Mining Methods.—The mine is developed by a coal gangway driven in the seam for a distance of about four thousand feet. Every three hundred feet, a pair of inclines are driven up the pitch, from which rooms are turned off every forty feet at right angles to the inclines. The room necks are twelve feet and the room widths twenty-four feet wide. When the rooms have been advanced one hundred and fifty feet, the pillars are taken out in retreat. In this way a high extraction of coal is secured.

The coal is undercut to a depth of six feet by means of radial machines. The cutting is done in the roof immediately above the coal in a soft stratum of 'cap-rock'. Two holes are drilled near each rib and sometimes a centre shot is used. Slow speed 'permitted' explosives are used for blasting. Compressed-air hoists are used for haulage on the inclines. Main and tail rope haulage is used on the main gangway.

Two grades of coal are made. Lump, over a $1\frac{1}{4}$ -in. screen, and coal through a $1\frac{1}{4}$ -in. screen, the latter being sold as slack. The lump coal is used for domestic purposes and the slack is sold as a high-grade steam coal.

The mine has a well equipped power plant, and the housing accommodation for the employees is of a high standard.

Taber District

The Taber coal field belongs to the Belly river formation but the Lethbridge and Taber coal basins are not connected; each forms a separate basin.

The coal seam mined in the vicinity of Taber is somewhat similar to that worked at Lethbridge, but has a rather higher ash content.

The mines are situated near Taber, a town on the Canadian Pacific railway, thirty miles east of Lethbridge. The coal seam, 4 feet 6 inches in thickness, is comparatively flat, and lies about one hundred feet beneath the surface.

The system of mining adopted in this field is room-andpillar and longwall. Electric coal cutters are used for undercutting.

The coal is screened and picked at well equipped screening plants, and the sizes marketed are lump, stove, pea, and slack.

Coalspur District

This district embraces the area contiguous to the Alberta Coal Branch line of the G.T.P. railway (now a part of the Canadian National Railways system) which runs from Coalspur to Lovett. The construction of this branch line was governed, primarily, by geological conditions, as the railway follows the valley of the Little Pembina river, which has cut its way along the strike of the coal measures.

A number of coal mines have been opened up along this railway, the most important producers being open-strip mines. The Sterling Coal Company is the largest producer. There is very little overburden and, as this is stripped off, the coal is loaded by means of steam-shovels. Two seams, originally forming the flanks of a small anticlinal fold, have been faulted and brought together to form a deposit of coal some one hundred and seventy-five feet in thickness. The coal is comparatively soft and is well adapted to steam-shovel methods of mining.

The coal is screened and prepared before loading into railway cars.

There are other, somewhat similar, deposits being worked by the Coal Valley Coal Company and the Blackstone Coal Company. At the Foothill Collieries, situated in this area, the coal is mined by ordinary underground methods.

Saunders Ridge Coal Co., Ltd., Mercoal:

This mine is situated at Mile 5, on the Mountain Park branch of the Canadian National railways, and is the only mine working on that leg of the coal areas.

The seam mined is the lower section of what is known as the Val d'Or seam, and the section mined is:

Roof Coal0 ft.	6 in.
Top Coal	6 in.
Bone0 ft.	1 in.
Clay 0 ft.	9 in.
Bone 0 ft.	
Coal	10 in.

The seams dip to the northeast at an angle of 34 degrees from the horizontal, and the opening is by means of slopes driven on the full dip of the seam.

Three slopes are in use, two being utilized for ventilation and one for hoisting purposes. They are driven a distance of 530 feet and two pairs of entries are set off, one pair at 250 feet and the other at the bottom of the slopes. These entries are driven east and west on the strike of the seams. The method of work adopted is room-and-pillar. Longwall has been tried, using an electric coal-cutter and conveyor and good results have been obtained. This system is being continued with the expectation of its being successful.

The entries are driven in pairs 12 feet wide and 9 feet high, the pillars between entries being 35 ft. to 60 ft. thick. Rooms are on 60-foot centres, the rooms being driven 27 feet

wide, with 33-foot pillars between.

The mining conditions are very hard and radial machines are used, cutting being done in the centre clay and the bone afterwards being removed. The rooms are driven up the pitch and the coal run down in chutes and loaded on the main entries.

The coal mines in large lumps and is finding an increasing market as a domestic fuel. It is low in moisture and stores well.

The boiler plant consists of two Scotch marine boilers, of 150 h.p. each, with feed water heater, injectors, and feed pumps. A fire pump is also installed for fire protection. For power purposes, an 185 k.w. generator direct-coupled to a Robb engine is used, and two straight-line air compressors are in use for supplying the radial coal cutter and jackhammer drills. Hoisting from the main slope is done by an Ottumwa hoist driven by an electric motor of 85 h.p. This machinery is all housed in a fire-proof building with corrugated iron roof.

Three cars are hoisted in a trip, each car having a capacity of $1\frac{1}{2}$ tons. The trips are hauled on to a trestle and lowered to the tipple, where the empty trip is picked up. The loaded cars are raised by means of a self-dumping cage to dumping level and dumped into a receiving chute from which the coal passes over a bar screen to the weight pan or basket. An electric hoist, driven by a 65 h.p. electric motor, is used for hoisting the loaded car and cage.

The coal is passed from the weight basket to a balanced Marcus screen where the cleaning and sizing is done. This screen is 50 feet long and 5 feet wide and is driven by a 40 h.p. electric motor. Sizes from $1\frac{1}{2}$ in. up are separated on the Marcus screen, and are made in egg, single and double screened sizes and loaded into box-cars. The larger sizes are loaded by means of an Ottumwa box-car loader of the pusher type, which is steam-driven. The smaller sizes, from $1\frac{1}{2}$ in. down, are passed from the Marcus screen to a bucket elevator which delivers the coal to a rotary screen placed over a large storage bin. The rotary screen further separates the small coal into nut and slack. The rotary screen and bucket conveyor are driven by a 10 h.p. motor. The tipple is of frame construction.

The following is the standard sizing at the mine:

D. S. lump	from 3 in. up.
S. S. lump	\dots from $1\frac{7}{8}$ in. up.
Egg	from $1\frac{1}{8}$ in. to 3 in.
Nut	from 1 in. to $1\frac{1}{8}$ in.
Slack	from 0 in. to 1 in.

All the large sizes of coal are weighed at the mine on a Fairbanks-Morse track scale of 100-ton capacity.

Ventilation is effected by a Sirocco fan of 50,000 cubic feet capacity, belt-driven by an electric motor of 15 h.p.

A well equipped machine shop with hammer, drill, lathe, power saw, welding equipment, etc., is provided, so that repairs can be made on the ground when necessary. There is also a wash-house, with showers and lockers for 120 men.

The Camp consists of twenty-four 3, 4, and 5 roomed cottages and other buildings, such as hospital, boarding house, bunk houses, hotel, store, and pool room; and a school to accommodate 40 children. A number of workmen have houses of their own.

The output from this mine during the past three years has been: 1924, 12,398 tons; 1925, 56,542 tons; and 1926, 66,987 tons. There has thus been a steady increase, and a large market is expected to be established by this Company on the Prairie and in British Columbia.

A conservative analysis of the coal follows:

Moisture 5 per	cent
Ash 7	4.6
Fixed carbon51	61
Volatile37	44
_	
B.t.u11,500	1

EDMONTON FORMATION

The coals of the Edmonton formation have a widespread distribution in Alberta, the area presumed to contain workable seams being estimated at 27,000 square miles.

The principal coal fields being worked in this formation are the Drumheller, Edmonton, and Pembina-Wabamun fields. In addition, coal is also mined at a number of isolated points, as in the Big Valley, Carbon, Three Hills, and Camrose districts.

Drumheller Coal Field

The Drumheller coal field, from the standpoint of production, is the most important domestic coal field in Alberta. Production has increased steadily and at a rapid rate from less than 15,000 tons in 1912 to nearly one and one-quarter million tons in 1920.

The Red Deer valley cuts through the entire thickness of the Edmonton formation, within which, in the strata exposed in the Drumheller district, a number of coal seams are irregularly distributed. The number and thickness of the coal seams and of the intervening sediments is given in the following table:

Coal Seams and Sediments, Drumheller District*

	Thick	ness
	Coal	Sediments
Seam No. 10 (highest)	1 .to 2 feet	40
Seam No. 9.	0 "1 "	40 to 55 feet
Sediments	0 1	10 " 20 "
Seam No. 8	0 "4 "	00 11 120 11
Seam No. 7.	1 " 6.7 "	80 " 130 "
Sediments		20 '' 28 ''
Seam No. 6	0.5 " 3 "	68 " 75 "
Seam No. 5	3.5 " 5.5 "	00 15
Sediments		8 " 20 "
Seam No. 4	0.5 " 1 "	7 " 10 "
Seam No. 3.	0.5 " 1 "	. 20
Sediments	10114 11	5 " 12 "
Seam No. 2	1.8 " 4 "	25 " 50 "
Seam No. 1	6 "7 "	
Sediments	0.5 " 2 "	30 " 50 "
Seam No. 0	0.5 2	90 " 100 "
Total	15 " 37 "	383 " 550 "

^{*&}quot;Geology of Drumheller Coal Field, Alberta", by John A. Allan; Third Report on the Mineral Resources of Alberta, 1921.

At present, four of these seams are being mined, namely, Nos. 1, 2, 5, and 7. However, the most important operations are confined to No. 1 or 'Drumheller' seam, and No. 5 or 'Newcastle' seam.

Drumheller Seam:

The principal mines operating on this seam (in order from west to east) are the Monarch, Midland, Scranton, Western Gem, Drumheller, Rosedale, Star, Yoho, Moonlight, Shamrock, Rosedeer, Western Commercial, Jewel, and Excelsior.

The seam occurs as two benches, the upper and lower, separated by from six inches to one foot eight inches of bone and high-ash grey coal. The usual practice is to mine the top bench only, which contains from four to six feet of clean coal. The roof is shale, and the floor is undulating. The mines are apt to be wet in places.

The seam lies from thirty feet to one hundred and eightyfive feet below the river valley, and is opened up by vertical shafts or slopes. There is a slight general dip to the west. The method of mining adopted is room-and-pillar. The coal cutters used are electric shortwall, compressed air radials, and punchers.

The coal is of high grade and will stand handling and stocking. It is prepared for market at well equipped screening plants where a number of grades are made.

Power is supplied by independent mine plants and by electricity from the central power plant.

Newcastle Seam:

The principal mines operating on this seam are the Newcastle, A. B. C. (Alberta Block Coal Co.), Premier, Atlas, Newcastle Junior, Hy-Grade, Elgin, Gibson, Midwest, Superior, and Western Gem.

The seam varies in thickness from three and a half feet to five feet five inches, but the average thickness where mined is about four feet eight inches of clean coal. There is one band of bone varying from a mere parting up to a maximum of twelve inches, but in most places this band measures less than three inches.

Electric shortwall machines are used for undercutting the coal. The roof is sandstone, and less timber is used here than in the Drumheller seam. This seam also is comparatively dry.

The coal is screened and prepared at well equipped screening plants, but on account of there being less bone in the coal it does not require the same amount of hand picking as does that of the Drumheller seam.

Markets:

Drumheller coal is sold, principally for domestic purposes, throughout the four western provinces. During the strike in the hard-coal mines in the United States, some Drumheller coal was shipped to North Dakota.

The mines are equipped to produce three times the tonnage of coal that the present market can absorb, and it is clear that, in order to secure efficient operation and cheaper coal, it is necessary to greatly extend the present market.

Edmonton District

The coal mines in the Edmonton district are situated in the vicinity of the city of Edmonton. This is an important field, and the production in 1922 was 550,000 tons. This coal is used for domestic and power purposes in the city of Edmonton, and a considerable tonnage is sold for domestic purposes throughout the four western provinces.

Two seams are worked, one approximately four feet in thickness and the other five feet three inches. The seams are practically flat-lying, and ninety per cent of the mines have been opened up by vertical shafts. About twenty-three feet of sediments separate the seams.

The coal is undercut by electric shortwall and compressedair coal cutters. The haulage is principally by horses, although main and tail rope haulage is used on some of the main roads.

The coal is screened and prepared at well equipped screening plants, and is marketed in the following sizes: 3-in. and 6-in. lump; 3-in. and 6-in. egg; nut; nut slack; pea; and pea slack.

Pembina-Wabamun District

This area is situated on the main line of the Canadian National railways, some fifty to seventy-five miles west of Edmonton. There are only two mines operating in this area: the Lakeside mine at Wabamun, fifty miles west of Edmonton, and the Pembina mine (North American Collieries), at Evansburg, seventy-five miles west of Edmonton.

The coal seams at these mines have not been correlated with one another, but it is believed that the Wabamun seam is higher in the measures than the seam worked at Evansburg.

The Pembina mine is the largest in the field, and is one of the best equipped domestic-coal mines in Alberta. The mine is opened up by a shaft three hundred feet deep, and is completely electrified; electric shortwall machines are used for cutting the coal, electric storage-battery locomotives for gathering, and trolley locomotives are used on the main roads.

The tipple is well equipped with shaking screens and picking belts for sizing and preparing the coal for market. There are also at the mine well equipped machine shops, carpenter shops, etc., for doing ordinary mine repair work.

PASKAPOO FORMATION

The principal mines operating in this formation are situated in the Tofield district. One seam only is being worked, which averages about six feet in thickness. The seam is flat-lying, and where it is worked the cover is about sixteen feet in thickness. The over-burden is removed by steam shovel and the coal worked as open strip. The coal is used principally for domestic purposes.

In the preparation of this paper, free use has been made of material contained in reports published by the Dominion Department of Mines and by the Scientific and Industrial Research Council of Alberta. The writer also wishes to express his thanks to the coal operators who have supplied him with information respecting their operations.

Ed. Note: The mine plans and photographs to illustrate Mr. Dick's paper were unfortunately destroyed in a train wreck and fire, and it has not been possible to obtain duplicates of many of them.

SECTION 3.—THE LIGNITES OF SASKATCHEWAN

By R. J. Lee (Member, C. Inst. M. & M.)*

(Jasper, Alta., Meeting, September 19th, 1927)

INTRODUCTION

There are two coal-bearing formations exposed in the Province of Saskatchewan. The higher is of Tertiary age, and probably comparable to the Fort Union group of North Dakota. The Tertiary beds cover a wide extent of country in the southern part of the Province, where they form the lower portion of the Cypress hills, the greater part of the Wood Mountain plateau, and the belt of elevated land forming the northern extension of the Missouri coteau. Eastward they form a shallow synclinal basin in the Souris River valley, and occupy some of the higher country to the north. Outliers containing some of the lower beds may occur in portions of the country to the east, but so far these have not proved of value as good coal lands.

The lower coal-bearing horizon is the Belly River division of the Cretaceous. This formation resembles very closely the Tertiary, in colour and in character of its fossils, but is separated from it by Cretaceous marine beds.

Tertiary (Fort Union) Coals

Throughout the hilly country of southern Saskatchewan, Tertiary beds are exposed. The underlying measures, outcropping in coulees and along the banks of creeks, contain lignite seams, which are mined in a small way, for the use of settlers, on fifty-four different locations between the Souris coal field and the Alberta boundary. The seams vary in thickness from $3\frac{1}{2}$ feet to 15 feet, the average working height being around 8 feet. All coal seams which are now being operated in the Province are of this age.

^{*}Supervisory Engineer's Division, Northwest Territories and Yukon Branch, Department of the Interior.

Cretaceous (Belly River) Coals

The rocks of the Belly River formation are exposed in the western part of the Province. A few isolated exposures of coal appear in the upper part of the formation, ranging from a 4-foot seam near Sastatchewan Landing, on the banks of the South Saskatchewan river, to exposures further north, where a maximum of 8 feet has been reported. The eastern extension of the formation probably thins out under the marine beds which occur in that direction. Southward, the Belly River formation is overlain by Upper Cretaceous shale, as has been proved in a borehole at Maple Creek, where, at a depth of 196 feet, the 4-foot coal seam was struck. second seam 7-feet thick was encountered 100 feet below the first, so that although the upper seam may not be exposed over much of the area occupied by the formation, the second may be expected to be fairly general in the southern part of the Province. Seams found in borings or natural exposures have the following thickness: 8 feet at Brock, 2 feet at Kerrobert, 8 feet at Salvador, and 4 feet at Unity.

Along the western boundary of the Province, the area covered by these measures extends from Walsh (20 miles west of Maple Creek) in the south to about 18 miles north of Maklin in the north. The eastern boundary is marked approximately by a line from Walsh to the South Saskatchewan river, about 18 miles east of Saskatchewan Landing, thence to White Bear lake and the vicinity of Perdue, Wilkie, and Swinbourne, where it turns west to the Alberta boundary—an area roughly estimated at 1,500 square miles (See accompanying map, Figure 45). There are indications that the distribution of these measures may extend beyond the area as here defined, but unfortunately careful records have not been kept by well drillers, and in some cases the information obtained may not have been made public.

HISTORICAL

The earliest record of the occurrence of coal in the Province of Saskatchewan is found in the report of Dr. Hector and Captain Palliser, of the Palliser expedition, in August, 1857. This deals with investigations made in the Souris valley.

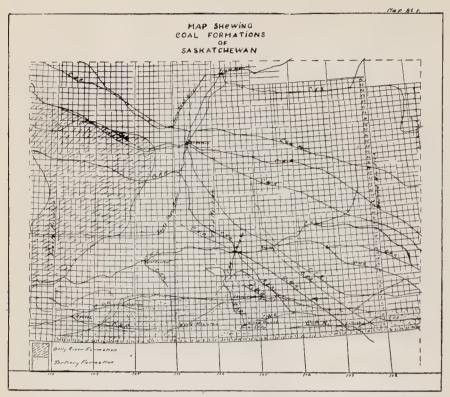


Figure 45.—Map showing coal formations of Saskatchewan.

It appears that these pioneers were induced to make a trip into the valley after hearing the Indians and half-breeds describe the quantities of coal that outcropped in the banks of the coulees, and along the creeks of the valley.

The next record is that of Dr. G. M. Dawson, in 1875, written at the time he was attached to the International Boundary Commission. In 1880, Selwyn conducted boring operations between the Souris valley (Saskatchewan) and Turtle mountain (Manitoba), and in 1903 D. B. Dowling reported on the geology of the Souris district.

In Willow Bunch district, coal has been mined in a small way for over 50 years. A Mounted Police barracks established at this point in the early 'seventies used coal mined in the vicinity. There are still residing in the town of Willow Bunch old half-breeds who can recall the mining of coal along the banks of Willow Bunch lake, about the year 1872.

It is not definitely known at what date coal was first mined in the Wood Mountain district, but as it was an Indian camp, and as the coal seams outcrop in many places, it is assumed that coal would be used in this district as early as in Willow Bunch, as intercourse between the two points was common and frequent.

In the Cypress district, there are no records of the date coal was first mined. It is, however, believed that coal mined in the vicinity of old Fort Walsh was used by the Mounted Police, stationed at this point about the year 1874.

Commercial production of coal began in 1880, when the late Mr. Hugh Sutherland shipped his coal by barge down the Souris river and other waterways to Winnipeg. The late Mr. Robert Hassard, Sr., was another of the early pioneers of the mining industry in the Province. Among the present operators there is one connecting link with the mining history of the past in the person of Mr. W. L. Hamilton, of the Crescent Collieries, who began mining operations in the Province in the year 1891.

COAL MINING AREAS

For convenience, the mining areas in the southern part of the Province have been grouped in four districts, according to geographical locations, as follows:

- (1). Souris District, comprising Townships 1 and 2, Ranges 6, 7, and 8, west of the 2nd meridian.
- (2). Willow Bunch District, including all the mines north and west of the Souris district and east of the 3rd meridian.
- (3). Wood Mountain District, including all the mines for ten ranges west of the 3rd meridian, and south of the C.P.R. Weyburn-Foremost line.
- (4). Cypress District, including all the mines west from the Wood Mountain area to the Alberta boundary. Properly speaking it commences at Shaunavon, in Township 8, Range 18, and continues fairly uniformly to the 4th meridian, coal being mined both north and south of the C.P.R. right-of-way.

Souris District

This is the principal coal-producing district of the Province and is served by both the Canadian National and Canadian Pacific railway systems, twelve mines having railway connection. Coal is mined at Estevan, Bienfait, Taylorton, Shand, and Roche Percee. The combined capacity of the mines

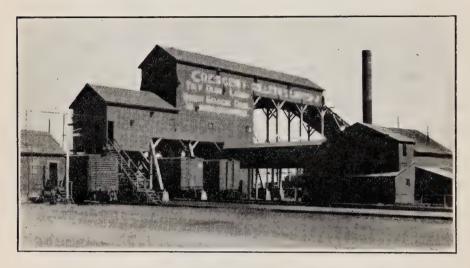


Figure 46.—Crescent Collieries, Limited, Bienfait, Sask.

in this division is around 5,000 tons per day, and they account for 96 per cent of the present output of coal in the Province. Most of the large mines are equipped with shaker screens, box-car loaders, and other facilities for the preparation and handling of coal, much improvement having been made in this respect within the past few years. Five of the mines use electric power-plants, whilst two purchase their electric power from the town of Estevan. Electricity is therefore used extensively for coal cutting, pumping, haulage, and of course for lighting and other purposes. Of the operating mines, six are worked by shaft openings, three by slopes, and the others by means of adit drifts driven in from the outcrop. On the large shaft mines, self-dumping cages are used. Two companies have their own locomotives.

The usual method adopted for working the coal is the double entry system, with rooms and pillars working out the panels. In several instances longwall mining has been tried,



Figure 47.—Bienfait Collieries, Limited, Bienfait, Sask.

but so far no general successful application of this system has been established. This is probably due to the fact that there is no supporting solid rock formation between the roof of the coal seam and the surface to take the dead-weight on the workings when large areas of coal are extracted. There has not, however, been any great difficulty experienced in working rooms to a width of 25 feet, when sufficient timber has been maintained. No inflammable gas has been met with in the field, and therefore carbide lamps are used throughout all the mines, and the coal is blasted by gunpowder.

There are four coal-bearing horizons, all within a thickness of 125 feet of strata, and within 150 feet of prairie level. These carry the lower, or principal, seam and three other seams. Each of these horizons has in places coal of workable thickness; also, each one has unworkable areas. The four seams have not been found together at any one point (See Figure 48).

No. 1 seam.—This seam is found only at the highest elevations to the south and east of the Souris river and Short creek. It has a probable areal extent of 20 square miles, with an average thickness of 4 feet. This seam is worked in one mine only, and is not considered of great importance.

No. 2 seam.—Occupies the same area as No. 1 seam but extends further northward, outcropping on the slopes of the river valley and in coulees. It covers an area of approximately 30 square miles, with an average thickness of 5 feet.

No. 3 seam.—This seam is more extensive. It is found at a low level near Roche Percee, and in most places below the Short Creek valley but it pinches out to the east, and in some places to the west it is represented only by thin streaks. It covers an area of approximately 90 square miles, with an average thickness of 6 feet. It is often classed as the 2nd workable seam.



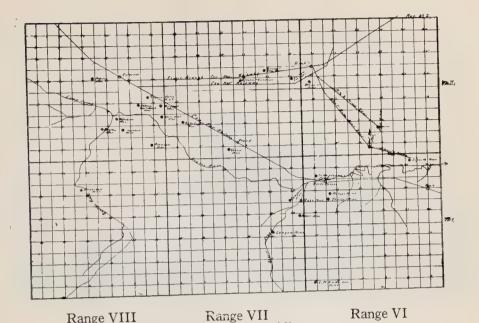
Figure 49.—Manitoba and Saskatchewan mine, Bienfait, Sask.

No. 4 seam.—This is the most important seam in the district. It is the one from which all the coal is mined in the Bienfait-Taylorton area. It is also mined at Shand and Estevan, with large outlying areas mined in a small way at Long Creek. It is often classed as the 3rd workable seam.

No seams are mined below No. 4, although there are in the Souris district four others with a wide distribution. These, however, are all smaller, and at considerable depths, and considering the enormous reserves in the four upper seams it is not thought that any of the lower ones will be operated for many years to come. (See accompanying maps, Figures 45 and 50).

Nature of the Lignite:

The lignite as mined carries a high percentage of water, and slacks readily on exposure to air. This disadvantage does not allow of storage during the summer months, and as



West of Second Meridian
Figure 50.—Souris Valley Coal Field.

a consequence most of the mines are slack during that season of the year. An average analysis of the coal as mined is as follows:

Moisture	33.0 per cent
Volatile matter	27.0 "
Ash	7.0 "
Fixed carbon	33.0 "
Sulphur	0.5 "
- Dupling VVVVVVV	
R t 11 per lb	6,500-7,000

The present market for the coal for power purposes is extending greatly, especially in Manitoba, where the coal is competing successfully with United States grades, owing to the comparative cheapness with which it can be mined. The short rail haul also gives it the advantage of low freight rate. For heating and domestic purposes it is also gaining ground. The output for the years 1906 to 1925, inclusive, is shown in the chart, Figure 52.

Relations between the operators and miners are of the friendliest, and whenever any differences do arise they get together and settle the matter amicably, without resorting

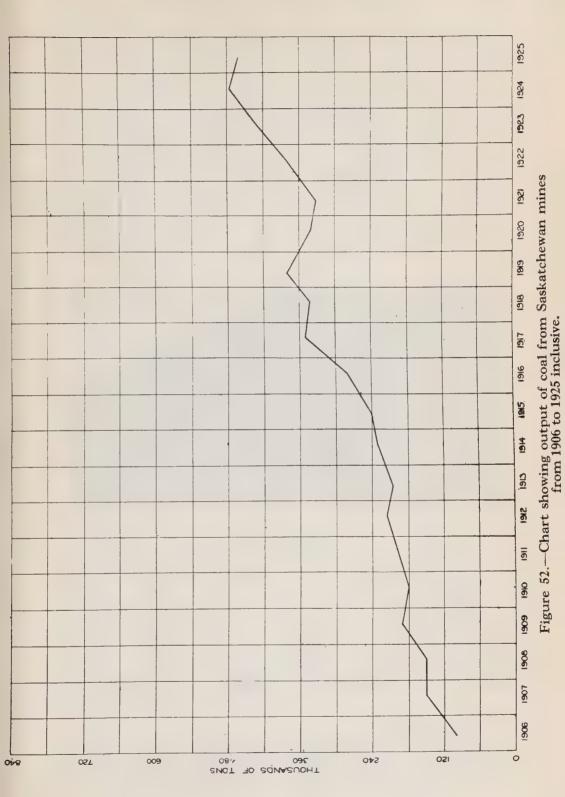


Figure 51.—Tipple and head-frame. Western Dominion Collieries, Taylorton.

to the costly business of strikes. In this respect the district holds the unique record of never once being tied up with a strike since commercial operations commenced in 1880. Some of the miners have been working at the same mine for 15 and 20 years, and are being followed by their sons. In many cases the elder men have saved enough to place them in comparative security against want. At the largest mines, the owners have provided their employees with all modern conveniences. such as electric light, water, and the option of telephones. The mining towns are provided with good schools, churches, adequate stores, and for the purpose of amusement there are motion picture theatres and community halls. miners' dwellings are suitably grouped, and the many fine gardens are evidence that the people grow a large proportion of their own vegetables. Practically every family has a motor car, and many radios are installed.

Willow Bunch District

Travelling west from the Souris coal-field, the first mines are met at Neptune, in Township 4, Range 16. From this point on the territory is all coal-bearing, and mines are operated (or have been operated) on approximately forty different



locations on the east side of the 3rd meridian. The furthest point north where operations are conducted is in the vicinity of Mitchellton, on Section 28, Township 10, Range 28, west of the 2nd meridian, about forty miles south of Moose Jaw. The seams are thinner here than elsewhere in this division, the average being about 4 feet. Travelling south from Mitchellton, towards Readlyn and Verwood, there is a slight tendency to increase in thickness of the seams. The same feature holds on the south side of lake Willow Bunch, and at Hart and Harptree they are around 12 feet thick. From this vicinity south to the International boundary, they decrease in thickness, and at Big Muddy Post, in Township 1, Range 21, the seam is 6 feet.

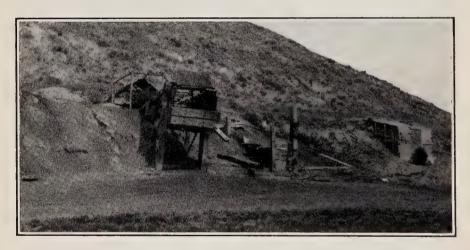


Figure 53.—Verwood No. 1 mine, operated by C. K. Sjdoin. Verwood, Sask.

Analyses of coal from several mines show that the seams are of slightly lower grade than the Souris lignite. It must be explained, however, that none of the mines have great development, and it is for the most part unfair to compare samples from this district with samples from the Souris district, where development has been carried on continuously for many years. With one exception, the mines are drift mines, and the coal seams are generally found outcropping in a coulee or along the lake shore.

The mines are mostly operated by farmers in a small way during the winter months, and as a rule but three or four men are employed at each; although sometimes as many as ten men are engaged for a short time at one or two of the principal mines. Approximately eight mines are open all the year round.

No coal is shipped from this district. The whole output is used locally by settlers, and it is difficult to imagine how this vast territory could have been settled if coal had not been easily obtainable. During the past year, however, four new branch railways were projected through this part of the Province, as follows:

C.P.R. from Assiniboia south into the Fife Lake district.

C.P.R. from Amulet travelling in a northwest direction.

C.P.R. from Broomhead travelling west.

C.N.R. extension of the Radville-Bengough line to Willow Bunch.

With the laying of these lines new towns sprang up practically over-night. What effect the increased activity will have on coal mining development of this large area it is yet too early to predict, but an opportunity is now presented such as was never before available in this district.

Wood Mountain District

Coal is being mined at fourteen points in the Wood Mountain district: at Maxtone, on Section 21, Township 6, Range 1, west of the 3rd meridian, the coal seam is 8 feet thick; on Section 16, Township 4, Range 4, and also on Section 17, Township 3, Range 8, west of the 3rd meridian, the seam is 6 feet thick. All the mining operations are conducted on a small scale. At Maxtone, the seam is worked by a shaft opening and a small steam plant has been installed. At Quaintock, a shaft opening is also used but the coal is hoisted by means of a gin driven by a horse. At two places the opencut method of mining is adopted, and the other mines are worked by means of drifts in from the outcrop.

The coal is used exclusively by settlers and, as in the Willow Bunch district, it has been of great benefit to the people who opened up this part of the Province. No railway branch lines pass through this district yet. It is,

however, expected that the C.P.R. eastern extension from Consul will meet their Broom Head western extension in the southern part of the area.

Analyses of samples taken from this district average about the same standard as those taken from Willow Bunch area, but, as already pointed out in connection with the latter, comparison of such samples with coals mined in the Souris district has little real significance. No extensive development has taken place, and since, where the coal is mined, there is as a general rule very little overburden, most of the coal so far mined has been from outcrop.

Cypress District

In this district, mining is carried on in the vicinity of the following towns: Shaunavon, Dollard, Southfork, Eastend, Ravenscrag, and at a few points in the Cypress hills. Shaunavon, the seam averages 12 feet on Section 3, Township 9, Range 18, west of the 3rd meridian, while on the south side of the town, where coal was first mined, the average thickness of the seam is around 6 feet. In the Dollard area all mining operations are by stripping or open-cut method, the coal seams averaging 7 feet, with from 5 to 10 feet of cover where operations have been carried on. At Southfork all mining has been by the open-cut system; the dimensions of the seam and other conditions are similar to those around Dollard. At Eastend, the coal seam is somewhat thinner, and mining in a small way is carried on both by drift openings and by the open-cut method. The same conditions obtain in Ravenscrag, and in the Cypress hills, where the coal seams cross the boundary line into Alberta.

Coal has been mined from Township 1 to as far north as Township 8, and, as in other districts, it has been of great benefit to the settlers. No coal has been shipped from any point in the Cypress district, and no expensive plants have been erected. The quality of the coal around Shaunavon is somewhat similar to that of the two districts to the east. In the south and west portion of the area, the coal appears to be of slightly better grade.

SUMMARY

Up till the present time, the only coal shipped in the Province is from the Souris Valley coal-field, and its greatest market is for power purposes; although in the autumn and winter large quantities are sold for domestic use. vicinity of the mines no other coal is used for any purpose, and in the open market its field of competition with the better grade coals is only limited by length of haul. It is by far the cheapest coal sold in Canada, and within reasonable limits it can be more economically used for the generation of steam than any other coal. With the extensions of new branch railways into the Willow Bunch area, it is not expected that any new market will be gained to the west. On the contrary, it is reasonable to assume that whatever market has been held in this direction in the past will be lost. The coal will therefore have to find its market in the north and east. principally in the east.

In the Willow Bunch area, the future market will probably be in the middle south, with a tendency to expand to the north, as far as the main line of the C.P.R. The prospects for the developing and marketing of coal in this district are most

encouraging.

In the Wood Mountain and Cypress districts, no extensive development is expected in the near future. The present market for the use of settlers is likely to be maintained, and probably somewhat enlarged as the population of these districts increases. However, in the event of large industrial development of any kind in southern Saskatchewan, there is no doubt the large coal reserves contained therein will be utilized to advantage.

SECTION 4.—OUTBURSTS OF GAS AND COAL IN COAL CREEK COLLIERY, B.C.*

By Bernard Caufield (Member, C. Inst. M. & M.)**

(Jasper, Alta., Meeting, September 19th, 1927)

INTRODUCTION

Some months ago the writer was invited to take part in the discussion on a paper entitled "Outbursts of Gas and Methods of Working Coal Seams Liable to Them", written by Mr. George Roblings, and read before the South Wales Institute of Engineers, at Cardiff, on November 16th, 1926.

Mr. Roblings is interested in a colliery at Ponthenry, South Wales, that has been subject to outbursts since 1918, and in which there has been serious loss of life from these dangerous occurrences.

Arising out of the part taken in the discussion of the aforementioned paper, the writer was prompted to make these notes with the hope that they may be of sufficient interest to have them discussed at the Edmonton meeting of the Empire Congress, at which papers dealing with the coals of Western Canada will be presented.

Outbursts in coal mines are not uncommon, as there are many coal mining districts in the world subject to them. It is not the intention of this paper to deal with these occurrences at any length, outside of the Province of British Columbia.

There have been serious outbursts in this Province at Cassidy mine (Vancouver island), Morrissey mine, and Coal Creek No. 1 East mine, causing serious loss of life in the two first named collieries. Morrissey and part of Cassidy were closed down on this account.

In August, 1903, an outburst occurred at Morrissey which blew out 1,630 tons of coal and a large volume of methane. On October 14th of the same year another took place, ejecting 800 tons of coal, with much accompanying methane, and suffocating four men. After this, it was thought that the places, which were

^{*}Submitted as 'Contributed Remarks' on the papers 'Coal Industry of Western Canada'.

^{**}Superintendent, Coal Creek Mines, The Crow's Nest Pass Coal Company, Limited.

working three shifts per 24 hours, were advancing too rapidly and not giving sufficient time for gas to drain off, and the workings were put on a single 8-hour shift per day, and so continued until November, 1904, when a very serious outburst occurred discharging 3,500 tons of coal, millions of cubic feet of methane, and killing 14 men. After this last outburst the mine was closed and has not worked since.

The outbursts at Cassidy have caused loss of life on two occasions.

During the past ten years there have been over 300 outbursts in No. 1 East mine, Coal Creek, ranging in size from a few tons of coal discharged to several hundred tons, and each one accompanied by very large volumes of methane.

Both Morrissey and Coal Creek mines are situated in the Rocky mountains, and are opened by adit levels driven in to the mountain side (See Figure 8, page 236). There is considerable overhead burden on the seams, with loose ends forming the sides of the valleys.

No. 1 EAST MINE, COAL CREEK

No. 1 East mine, Coal Creek, was opened on the south side of Coal Creek valley in 1911. The method of work is the bordand-pillar system. Owing to there being from 2,000 to 3,000 feet of cover on the inner workings, only about 15 per cent of the coal seam is mined, with no pillar extraction. The main levels are driven on the strike of the seam, which is due south, and cross entries are driven east and west, the seam dipping eastwards about 10 degrees. Rooms, or stalls, are opened off the cross entries in pairs, having a 60-ft. pillar between rooms and a 200-ft. pillar between pairs. Entries are driven wide enough to take a 10-ft. cross bar of timber, and rooms wide enough for a 14-ft. cross bar. The pillar between rooms is cross-cut at intervals of 60 ft. and the 200-ft. pillar is cut through at intervals of about 200 ft. for ventilation. All rooms are driven parallel to the main entry. The distance between slopes and inclines is about 1,000 feet, and rooms are driven 800 feet, leaving a 200foot barrier-pillar between the finishing point of the rooms and the next slope or incline. (Figure 54).

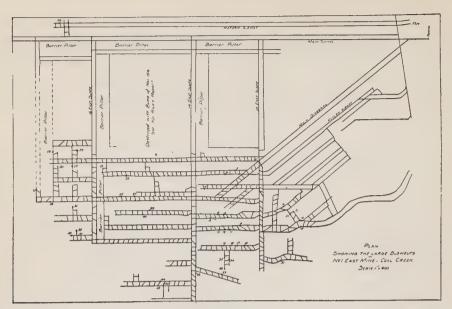
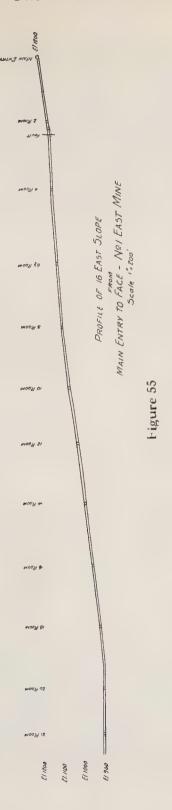


Figure 54.—Plan of No. 1 East Mine, Coal Creek, showing locations of the large blow-outs.

After the slopes are driven down about 3,600 feet, they flatten off and then start to rise again, thus forming a basin or syncline, in which faults and other irregularities in the formations have been met (See Figure 55).

All outbursts have occurred in the vicinity of this syncline, mostly on the slopes on each side of its central axis. The coal seam is of high-grade bituminous quality, running from 20 to 25 per cent volatile. There is no regular cleavage in the seam, and places work equally well in whatever direction they are driven.

There are backs and slips in the seam running in all directions, backs varying in thickness from a few inches to a foot or more. The cracks or slips dividing the backs are from a fine slit to half an inch wide and are full of fine dust and gas. The seam is from 15 ft. to 30 ft. thick, but owing to there being one, and sometimes two, bands of fine slickensided shale in the upper part of the seam, the bottom 10 feet only is worked. The coal is friable and soft, breaking readily when handled. There is, however, a band of fairly hard coal from 9 to 12 inches thick, about 10 ft. from the floor, and 2 ft. below the first band of rashings. When mining, which is all pick work, this hard band has to be broken through to give the top part of the seam a



chance to expand. Failure to do this contributes to the risk of outburst. The expansion action of the top part of the seam is destructive on timber.

The seam gives off from 3,000 to 5,000 cubic feet of methane per ton of coal mined. When the miners are cutting through the backs in the seam, gas issues so freely from the crevices between them that a flame safety lamp held close to the face will be immediately extinguished. Edison electric safety head-lamps are used by the workmen. Firebosses and bratticemen are the only persons allowed to use flame safety lamps. No electricity is allowed in the mine, and shooting is prohibited, except in rock work.

The number of places worked on a split varies according to the amount of gas given off. When the air-current from any district entering the return airway is carrying up to 2 per cent methane in the general body of the mine air, no more working places are allowed to operate on that particular split. At the end of each split there is a gas-testing station with a Burrell gas detector and a blackboard.

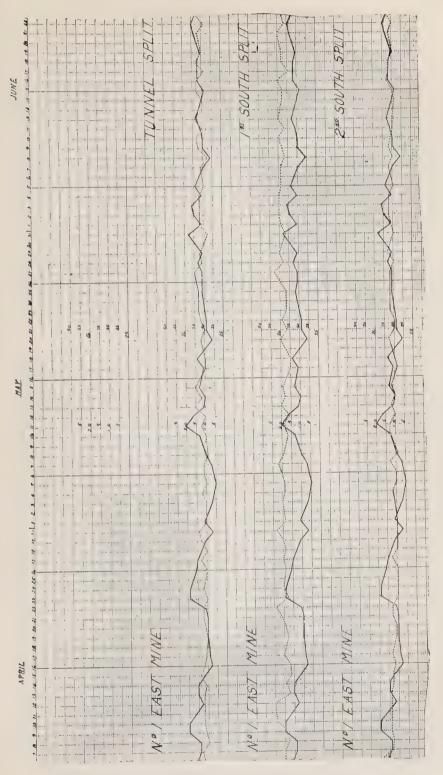
Tests of the mine air are taken several times a day and the result of each test marked on the blackboard. Burrell gas tests are recorded daily on a graph kept in the superintendent's office (Figure 56).

Mine air samples are taken monthly from each split, for analysis by the Government Mine Inspectors. The samples are correlated with Burrell gas detector and the Wolf flame safety lamp, and checked on the return of the analysis. The information obtained from these records is of inestimable value to the underground officials.

The first outbursts occured in this mine in the latter part of 1916, and there has been a continuation of them at intervals ever since. A detailed record is kept of all these occurrences. The most severe are marked on Figure 54.

Three of these occurrences are described in some detail below. They differ in some minor particulars, but in their general characteristics they are typical of all the outbursts so far encountered.





The Outburst of April 5th, 1926

A severe outburst of coal and gas occurred in No. 1 Left level, off 13 slope, No. 1 East mine, on April 5th, 1926, at 8.15 p.m. (See No. 33 on Figure 54). This place is being driven 10 ft. wide and it is running almost parallel with a fault-line between Nos. 10 and 13 slopes. The place was worked double shift only on the day of the outburst. The day-shift miners loaded about 16 tons, and the place to all apearances was in normal condition. The miners on the afternoon shift were loading their sixth car when the face started working, making noises, cracking and pounding. They immediately withdrew from the place, and when they had reached a point about 300 ft. from the working, the outburst occurred. They estimated that it was about 5 minutes from the first warning until the final burst. The shock or vibration of the outburst was felt all over the mine and on the surface, shaking all the buildings in the valley.

This place was the first one on the ventilating split, and the whole 'district' was immediately filled with gas. The district was examined at 9.30 a.m. the following morning. It was found that there was 2 per cent methane in the air-current from this split, with a quantity of 26,000 cubic feet of air travelling per minute. On examination of the place of outburst, it was found that a large quantity of coal dust, loose coal, and rock had been ejected. The dust was about 3 ft. thick in the cross-cut; it was very fine and dry, so much so that when one tried to grab and hold a handful, it would slip out between the fingers. The dust piles were wavy like sand dunes or drifted snow after a blizzard, and there was a thin coating of coal dust sticking to the timbers all over the district. In cleaning up the place, 400 tons of loose coal, 200 tons of rock, and 100 tons of dust were loaded out. It was estimated that over 2,000,000 cubic feet of methane was discharged (See Figure 57).

Two samples of gas were taken from this place, and they contained 92.2 per cent and 84.5 per cent of methane respectively.

The Outburst of November 5th, 1926

On November 5th, 1926 an outburst of gas and coal occurred in No. 2 cross-cut, 17 room, 16 slope of No. 1 East mine (No. 31 on Figure 54).

This place was being driven through the big pillar between pairs of rooms, for ventilation. As the pillar is 200 ft. thick it is customary to drive two places, cross-cutting them every 50 ft. but stopping one of them 50 ft. back, and just driving one place through.

The place of outburst had been working in tough coal for a distance of 40 ft. An outburst was expected on this account, as this is one of the characteristics preceding these occurrences. A drill hole 17 ft. long had been drilled in the face $3\frac{1}{2}$ hours before the outburst, and the place had been idle from Wednesday, 2.00 p.m., until Friday, 7.30 a.m. The miners had loaded two cars when the place started to work; then the usual warning, tap - tap - burr-r-r- bang, commenced, and a few seconds later the place burst out. A very large volume of methane was discharged with considerable force. A horse standing at the bottom of the place (100 ft. away) was knocked down. The whole district was filled with gas.

The writer made an examination of this place about 3 hours afterwards and found that the coal-face was not broken up, as is usually the case in these events, but was pushed out *en bloc* a distance of 7 ft. The track was within 6 ft. of the face before the outburst. The coal face pushed against the end of the track and curved the last pair of 20-lb. steel rails. A tape line was stretched from each end of rails, forming a chord, and the height of arc from tape to rails was 3 feet. (See Figure 58).

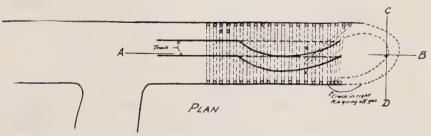


Figure 58.—Effect on track of outburst of November 5th, 1926.

Temperatures were taken. The temperature of air current on the intake side of the brattice was 57° Fah. The thermometer was afterwards placed on the face coal in a body of gas and left about 10 minutes. On reading it was 52°Fah. Gas was issuing freely, with a hissing sound, from a crevice on the right rib.

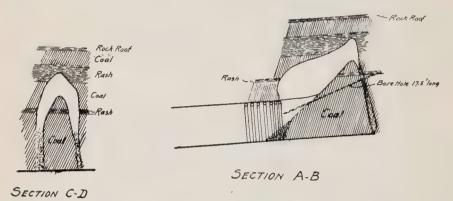


Figure 59.—Blow-out in No. 2 cross-cut, 17 room, 16 East district, No. 1 East Mine, Coal Creek.

The following morning, November 6th, the writer again visited this place and found the gas still flowing from the crevice. Some more temperatures were taken. The intake air behind the brattice was 54°Fah. The thermometer was afterwards placed in the crevice on the rib side where gas was coming from, leaving it 10 minutes. On withdrawal it registered 46° Fah.: a drop of 8° Fah. Estimated amount of gas discharged was 3 million cubic feet. This was checked over a period of 35 hours by taking readings with the Burrell gas detector in the return airway from this split.

Work was resumed in this place three days after the blow-After cleaning up the loose coal it was found that the working section of the face that had pushed out, had broken off the solid coal and was standing out, like a huge monument. in one piece that contained 42 tons (See Figure 59).

Altogether there were 85 tons of coal and rashings loaded out. Estimated tonnage displaced, from volumetric measurements, is about 50 tons. This is characteristic of almost all our blow-outs—the cavity left is smaller than the volume of the coal discharged.

After cleaning up, an examination was made of the solid face. There was nothing unusual about it. The coal on the left side was firm and solid. On the right side it was cracked and broken. Two boreholes were drilled; one on the left side 18 ft. long, and one on the right side 13½ ft. long. There was nothing unusual in the first hole; it was in solid coal all the way. The hole on the right side was in broken coal for a distance of 7 ft. Every foot or so the drill met a break and would advance quickly for a few inches at a time. From a point 7 ft. from the mouth of the hole the drill was in fairly solid coal up to the end of the hole, a distance of 13½ ft. It was noticed that the borings from this hole felt colder than the loose coal lying at the face. Temperatures were then taken. Mine air temperatures were 57° Fah. The thermometer was then fastened on the end of a long rod and pushed into the hole. On withdrawal it registered 50° Fah.; a drop of 7° Fah. Figure 60).

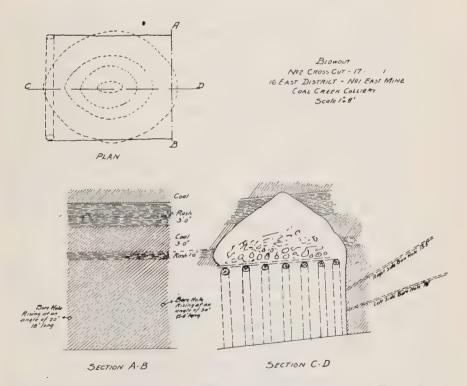


Figure 60.—Blow-out in No. 2 cross-cut, 17 room, 16 East district, No. 1 East Mine, Coal Creek.

Samples were taken of gas and coal from this outburst and sent to the U.S. Bureau of Mines, Pittsburgh station, for tests. The gas was found to have the following composition:

CO_2					,					6 1			0			, ,									۰		3.5
O_2									a		 ٠	٠								٠	0						0.0
N_2										a 1											۰	۰	D			å	0.0
CH_4									۵		 ø						۰							0			96.27
C_2H_6				٠,			٠.			• .					٠			w						۰			0.21
C_3H_8					10									,					a				۰	e	a		0.02
C_4H_{10}	ar	nd	1	niį	gł	ıe	r				a	ø		0 1									٠	a		۰	0.0

Two samples of coal were taken, and the results of the tests made on these are set forth in the accompanying table.

The Outburst of March 22nd, 1927

A large outburst of gas and coal occurred in No. 1 East mine, Coal Creek, on Tuesday, March 22nd, 1927, at 1.20 p.m. (See No. 46 on Figure 54).

This place and the parallel were working in the top section of the seam, having the rashing near the floor. The roof is a hard shale. The place was working single shift. The miners had loaded about 10 tons and the place, to all appearances, was in a normal condition. About two hours before the outburst the place gave a few light knocks. Nothing more was heard until a few seconds before the blow-out, when the face gave a crack, like the discharge of a rifle shot. miners stopped working and listened; they heard a grinding or grating noise. They decided to withdraw, and had only got a few feet away from the face, when the usual pounding and thundering noise commenced, and shortly afterwards the outburst occurred.

This place was working uphill and a cross-cut was being driven from the parallel place and was within 12 ft. of holing. This latter place had been hard for a time, until about one hour before the blow-out, when it became much softer, the coal in the face began to work freely and the roof became uneasy. This is a sure indication of an outburst.

The discharge of methane was so violent that it forced open the wooden trap door between the two places, and backedout against the intake current for a distance of 1,000 ft. It

Sample No. 1.—Coal taken from face of No. 2 cross-cut room, 16 East slope, No. 1. East mine, Coal Creek, B. C. Taken after an outburst of gas and coal on November 5th, 1926. Occluded Gases in Coal Samples.

	Gas	Gas pumped off	f uncrushed coal	coal.		Additional after grind	Additional gas pumped off after grinding in vacuo.	off	Total ga	Total gas pumped
	Ana (per c	Analysis (per cent)*	Cubic ce per 100	Cubic centimeters per 100 g. coal†	An: (per	Analysis (per cent)	Cubic ce per 10	Cubic centimeters per 100 g. coal	Cubic centimete per 100 g. coal	Cubic centimeters per 100 g. coal
	A	В	A	В	A	В	А	В	A	B
\mathcal{O}_2	24.3	19.2	39.6	39.6	13.4	12.6	6.7	6.7	46.3	46.3
63	0.0	4.4	0.0	9.1	0.0	1.2	0.0	0.5	0.0	9.6
H_4	52.0	41.1	85.0	85.0	42.6	40.0	21.4	21.4	106.4	106.4
$\mathbf{C}_{2}\mathbf{H}_{6}$	3.4	2.7	5.6	5.6	21.9	20.6	11.0	11.0	16.6	16.6
. 64	20.3	32.6	32.8	0.79	22.1	25.6	11.3	13.9	44.1	6.08
Total	100.0	100.0	163.0	206.3	100.0	100.0	50.4	53.5	213.4	259.8

CO present, less than 0.01 per cent.

Sample No. 2.—Coal taken from same place us No. 1, but back on rib side in hard knotty coal that usually precedes a blow-out of gas.

	37.5	12.1	88.2	17.0	126.0		280.8	
	37.5	0.0	88.2	17.0	80.2		222.9	
	5.5	0.1	6.6	6.6	11.4		36.8	
	5.5	0.0	6.6	6.6	10.7		36.0	
	15.0	0.4	6.92	27.1	30.6		100.0	
	15.3	0.0	27.4	27.6	29.7		100.0	
2000	32.0	12.0	78.3	7.1	114.6		244.0	
	32.0	0.0	78.3	7.1	69.5		186.9	
	13.1	4.9	32.1	2.9	47.0		100.0	
	17.1	0.0	41.9	3.8	37.2		100.0	
	CO ₂	O_2	CH_4	C"H,	Z		Total	

^{*} All analyses are calculated air-free under A; as found on analysis with air present, under B. † At 0°C. and 760mm. pressure.

was estimated that $3\frac{1}{2}$ million cubic feet of methane was discharged, and 400 tons of coal, 30 tons of rock, and 20 tons of of very fine coal dust.

After cleaning this place it was found that the outburst came from the solid coal on the left side and from the face. The coal was shattered and broken for a distance of 18 ft. forward, in the direction the place was going, and for a distance of 25 ft. in the solid on the left side, opposite the point where the crosscut would hole. The ejected coal and rock forced out most of the timbers, breaking three posts from the front of the face, and 13 posts on the right side rib, about 4 ft. up from the floor (See Figure 61).

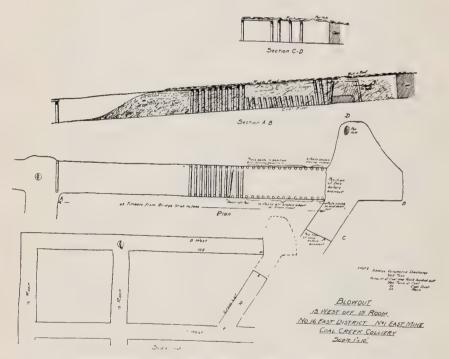


Figure 61.—Blow-out, 18 West off 15 room, No. 16 East district, No. 1 East Mine, Coal Creek.

The place was working on the face of the backs and the cross-cut, coming at an angle of about 60°, was partly on the end of them. After the cross-cut was holed through, the backs were parted from each other with the open crevices between, facing into the cross-cut. It was thought that, had the crevices been full of gas under high pressure, it would have discharged

into the cross-cut along the plane lines or cracks looking in that direction. Regarding the gas backing against the air current, this is a common occurrence in large outbursts.

THEORIES ADVANCED FOR THE CAUSE OF OUTBURSTS

The following theories have been advanced from time to time as to the probable causes of outbursts.

Depth.—Some authorities attribute these occurrences to the excessive pressure exerted on seams at great depth. In the conditions at Coal Creek, this theory is not supported by the fact of No. 2 seam being 200 ft. deeper than No. 1 seam, each working under the same mountain. No. 2 seam is equally as gaseous, and parts thereof more so than No. 1 seam, yet we never have any outbursts in No. 2 seam. This of itself is sufficient to disprove the depth theory.

The Cavity Theory.—This theory is based on the belief, held for a long time, that there were cavities or reservoirs of gas under excessive pressure pent up in the seams. Old miners, when referring to them, used the expression 'bags of gas' or 'pockets of gas.' Pressure of gas as high as several hundred pounds per square inch has been recorded in boreholes, and probably this fact has given rise to the cavity theory of pent up gases.

Anyone who has had experience with outbursts no longer accepts this theory, as there has not been a cavity of gas found anywhere in the form stated.

There have been over six miles of advance drill-holes bored ahead of the working faces during the past few years in Coal Creek, and in no single instance have they encountered any cavities or reservoirs of pent up gases.

In Belgium, it has been said of this theory: "No such pockets have ever been found, and none of the hundreds of blow-outs that have occurred in Belgium have ever disclosed empty spaces larger than could be accounted for by the quantity of solid material which is ejected with the gas". Moreover, the Ponthenry investigation proves that "long boreholes drilled in the seam itself failed to relieve gas pressure, or indicate any cavity whatever in advance of the working places".

Mr. James Ashworth's Theory.—Mr. James Ashworth advanced the following theory for the Morrissey outbursts: "Outbursts were attributable to the volatilization of light oil or spirit, which had been absorbed in patches of the soft coal, and on being released by the removal or thinning of the surrounding coal, became volatilized with accompanying violence. As layer after layer of saturated coal was blown off, the dust was carried away by the gas, and the outburst continued until the oil-saturated mass was blown away and the oil spirit volatilized". In another article at a later date, he states, of a Coal Creek outburst, that "the fine coal discharged had a decided greasy feel rubbed between the fingers, and this fact in itself would suggest that oil has some intimate connection with the phenomenal outbursts of gas".

Regarding this theory, the outbursts in Coal Creek do not support the contention that there is any oil or spirit present in the seam; nor has any greasy coal ever been found after any of the outbursts.

The Adsorption Theory.—This theory was advanced by Prof. J. Ivon Graham and Dr. Henry Briggs. In the discussion of Mr. Roblings' paper, Dr. Briggs states: "A few years ago Mr. J. Ivon Graham and myself, working independently on a number of porous materials possessing the property of condensing or adsorbing gas, both had cause to investigate the effect of pressure on that process. The results threw a flood of light on outbursts. Taking some Ponthenry outburst coal, I endeavoured to restore its status quo by forcing gas (first methane, then carbon dioxide) into gas cylinders into which the coal had been packed. The enormous capacity of the fine, loose, dry coal for either of these gases became evident; indeed for pressures of 75 lb. or 90 lb. per square inch, the coal was found to hold more methane, and considerably more carbon dioxide, than the equivalent volume without the coal. It is something of a paradox that, to store a maximum of these gases under pressure in any space, one ought first to fill the space with dried outburst coal, or, better still, with dried activated charcoal, and that the more the coal or charcoal is rammed into the space the greater is the volume of gas that can be pumped in also".

It was also found by experiment that dry coal has a greater affinity for water than has fire-damp, and if a practical method could be found to pump water under pressure into the affected area of a seam, it would go a long way towards eliminating outbursts.

Prof. Graham read a paper in England on "The Adsorption or Solution of Methane and other Gases in Coal, Charcoal, and other Materials", in which he reports the results of a long series of experiments in the adsorption of gases in coal and other materials. Prof. Graham says in part: "This phenomenon of the adsorption of gases by solids or the withdrawal in a selective manner of dissolved substances from their solution in the absence of, or apart from, any chemical action between the material and the gas, or dissolved substance, is generally termed 'adsorption' or 'solid solution'. Either of these are useful terms to cover a physical change which is not yet thoroughly understood, but which is now generally explained as one in which the molecules of the substance adsorbed are attracted to the surface, and particularly the internal surface, of the adsorbing material (coal, charcoal, etc.) and there momentarily retained through the exercise of intermolecular forces."

Mr. Graham by experiment showed that at 30°C. coal may take up at atmospheric pressure three times its volume of methane. On a suggestion by Prof. Haldane, Mr. Graham conducted an experiment with the object of ascertaining whether softened coal in lump form thoroughly charged with gas at high pressures, would, on suddenly releasing the pressure, fly into dust on account of the very high adsorbed pressure of gas inside the lumps. In order to ascertain this, 1/41b. of Ponthenry coal was placed in a stout metal shell, suitably fitted, and was connected to a cylinder of compressed carbon dioxide, up to a pressure of as high as 50 atmospheres, and the pressure maintained for 22 days. The apparatus was then disconnected and the pressure suddenly released; it dropped to atmospheric pressure in 8 seconds. On opening the shell not a trace of dust was found, but the coal continued to give off the dissolved gas for many hours afterwards. Several experiments along the same lines were made, with no different results.

They concluded that these experiments proved conclusively that the coal in lumps highly charged with gas does not disintegrate into dust when external pressure is suddenly diminished.

After these experiments, Dr. Briggs supported the theory that outbursts are due to the presence of large accumulations of coal ground to fine powder by earth movements, containing methane in large quantities under considerable pressure.

Mr. G. S. Rice, Chief Engineer of the U.S. Bureau of Mines, in his Report, "Bumps and Outbursts of Gas in the Mines of the Crowsnest Pass Coalfield", refers to the gaseous condition of this field and says in part: "Under the great pressure of the heavy covering, the Crowsnest coal in the process of mining is subjected to a squeezing action, which tends to crush and ground the coal particles one against the other. This may produce a condition which laboratory grinding would be analogous to. How occluded gas is held in the coal substance is an unsettled question, that is to say, is the coal substance so impervious that the gas is held in minute pores, like little bottles, only to be released when these are broken, or is it held by chemical bonds, so unstable that on a slight relief of pressure the gas is given off? Some physicists have contended that films of gas are held on the surface of the particles of coal under such tension that the gas is in compressed state equal to liquefaction. If so, when the coal bed is opened the coal near the headings and rooms may begin to release its contained gas".

Theory of Mr. G. S. Rice.—In the discussion of Mr. Roblings' paper, Mr. Rice gives the following theory of out-bursts:

- 1. Instantaneous outbursts are the result of geologic lateral movements produced by folding after a coal bed has been formed, and in some instances in recent geologic times.
- 2. Such lateral movements, possibly in different directions, at different times, have crushed the structure by a combination of stresses in certain places, now the seat of possible outbursts.
- 3. It is known beyond controversy that coal when crushed, even without heating, liberates a large amount of gas which in this case would be accompanied by some frictional heating.

- 4. It has been demonstrated in laboratory testing by both British and American investigators that coaly particles adsorb gases proportionately in some degree to the pressures of the gases. I assume that the gases are held on the surface of the particles in the spaces between dust particles, without reference to the gases already imprisoned in the microscopic cells.
- 5. It has been repeatedly observed that there are no pockets or recesses in places of outburst; in fact, those who have observed the effects have commented that there appeared to be much more material ejected than could have come from the outburst area. Moreover, bore holes which have penetrated into such areas have not found openings.
- 6. From the foregoing, I think it will be generally conceded that an outburst area is tightly filled with coal and dust before the outburst.
- 7. Certain of the French students of the question believe that prior crushing of the coal structure is not necessary to explain an outburst, but that it is due to local ground stresses. I am not able to see, if this is the case, why such outbursts do not occur in more mining districts. For example, in the Nanaimo field there is only one of many mines that experience this phenomenon. This is also true in the Crowsnest coalfields, where several mines adjacent to No. 1 have not had outbursts, although under similar surrounding conditions. It is also true in Belgian mines, and I think in French mines.
- 8. It is not a question of depth, at least below a depth of 750 to 1,000 feet, as is apparent if one studies the records of 137 Belgian outbursts assembled by Stassart and Lemaire.
- 9. Moreover, how can one reconcile the carbon dioxide outbursts in the south part of the Gard basin (the effects of one of which I observed) in the absence of notable amounts of hydrocarbon gases, if the outbursts were the result of the coal being suddenly fragmented under pressure, which must chiefly liberate hydrocarbons.

- It is my tentative belief, from the foregoing probable 10. facts, that outburst areas are those which have been produced by geologic stresses more or less localized, that have crushed the coal, and, that the gas liberated has not been able to escape from the immediate surroundings. Under this tentative theory, there may have been countless places similarly crushed, but the gas escaped through coal and rock jointings and fault places, during the countless centuries.
- 11. When the natural impervious shell is approached by mining, a point is reached where its strength will no longer hold under the great pressure of the gases.
- A bore hole into such an area does not give off a 12. large amount of gas for two possible reasons:
 - the fine dust from the crushing seals the walls of the bore hole in the same manner as fine silt will often clog the suction strainer of a pump.
 - the rapid expansion of the highly compressed gases (b) may cause, if the coal is sufficiently moist, the particles adjacent to the wall of the bore hole to freeze together.

In the observations here, an attempt has been made to find some relationship between the manifestations presented by these outbursts and the various theories advanced, and these efforts have resulted in the collection of a mass of contradictory Whatever theory is advanced should be consistent in principle in the majority of cases. As regards the adsorption theory, bore holes or working places have never encountered any portion of soft, crushed or powder coal. In the outburst on November 5th, 1926, referred to above, the face was pushed out en bloc and the large piece of coal broken free from the solid contained 42 tons. In five large outbursts this year (1927) in the months of January and February, no dust was given off. In each case the loose coal discharged was just like run-of-mine coal.

It is recorded of the last outburst in Cassidy, where two men were killed, that no gas was given off, and the bodies were recovered 15 minutes after the outburst. How would the adsorption theory explain these incidents?

After the blow-out of April 5th, 1926 in which 700 tons of loose material and some millions of cubic feet of methane were discharged, the writer made the following comments to the United States Bureau of Mines:

"I am not in sympathy with the ordinary explanations given as to causes of blow-outs, because:

- (1) I cannot see why a cavity of pent up gas under pressure does not dissipate itself in such a porous seam.
- (2) Why should parts of seam stop exuding gas previous to an outburst?
- (3) The seam gets hard, probably due to the fact that the gas stops exuding and thus is not actively assisting to break the coal previous to blow-out.
- (4) Miles of drill holes have not encountered any excessive gas pressures or cavities of gas.
- (5) The cavity left after a blow-out is not in the least commensurate with the amount of material displaced.
- (6) We have never met with any pockets of fine coal or dust, either with drill or in working places.

"Having these ideas in mind I am of the opinion that there are points in the mine of regional extreme pressure due to the heavy overburden and geological movements under stress, and when these pressure points are released they manifest themselves with explosive force: the forces breaking up the coal so violently as to disassociate the gas from the coal, separating the two."

Mr. Rice advised at that time that the writer's theory had been advanced previously by French engineers, although the writer had not heard of it at the time. His reason for advancing this theory was due to the fact that the outburst referred to was accompanied by earth shocks, that shook all the houses in the valley within a radius of over one mile. Large outbursts usually cause earth movements which are felt mostly in the mine; some of them on the surface.

In these British Columbia mines there are daily manifestations of the relief of pressure. In No. 3 mine, No. 2 seam, severe pounces occur at the coal face, mostly when cross-cuts have holed. These go off like cannon shots, throwing coal out of the solid and off the pillars. They rarely disturb the roof or timber, and do not give off gas. Again, there are days when the whole mine seems to be on the creep; the roof thudding, cracking, and renting over large areas. There are sometimes bumps that shake the whole valley, and yet do no damage to the mines. On other occasions the bumps are very destructive to underground roadways. It is these occurrences which give rise to the idea of physical pressure points being present.

Regarding the experiment of Prof. Graham in subjecting coal confined in a closed vessel to gas at high pressure for the purpose of finding if coal with high adsorptive pressure would fly into dust when the pressure was suddenly released, it is very interesting. The fact that it failed to do so, would tend to show that some other pressure was necessary to account for the fine dust accompanying outbursts, and further would lend support to the theory that physical stresses were present at or near the point of blow-out.

The adsorption theory gives rise to the idea that although there are not 'pockets' in the accepted meaning of that term, there are patches of soft fine coal charged up with gas under pressure, held by adsorption. If this were so, why should not a drill hole sooner or later discover such a condition? On the other hand, drilling would not relieve a condition of pent up strain or pressure caused by geological movement and compressive stress. Or is it possible that there may be a combination of physical stress and adsorptive pressures existing at the same time, which, when given a chance, would burst forth with the results noted.

The fact that outbursts usually occur from points where faults, swellies, pinches, and other geological irregularities are met, supports the theory of geologic movements.

SAFETY PRECAUTIONS AND SUGGESTED REMEDIES

Whatever may be the true cause of these occurrences, the means to avoid the dangers incidental to them are of far more importance and concern than the cause. Several measures have been adopted to safeguard the men in the mines subject to outbursts. An outline of some of these follows:

Boring Advance Drill Holes

This method is used for the purpose of relieving pent up gas pressures, supposed to be stored up in cavities or pockets in the seam. The results have been very disappointing, as they have never found the conditions expected. In Coal Creek there have been nearly 7 miles of holes drilled, and they have never encountered any pockets of gas, or fine, powdered or crushed coal, or high gas pressures.

A driller bores two holes about 20 ft. long per shift in places reported by the mine officials as likely to need attention. A record of each hole is kept, the driller having instructions to report as follows:

INSTRUCTIONS RE BORING HOLES

Make reports as concise as possible, using the following terms:

Date. - Give date on morning of which report is made.

Place. - Give location and place, as: 24 room, 6N. X-cut.

Holes. - Number of holes drilled in place designated.

Length. - Give length drilled in feet.

Gas. - Use the following terms:

None. — When no increase in gas cap at mouth of hole.

Light. — When increase in gas cap shows, giving size.

Steady. — When light pressures can be felt.

Strong. — When gas is blowing from hole noticeably.

First Flow. — Give distance drilled in feet at which flow of gas is first noticed.

This would probably only apply when 'strong' or 'steady' as above are used.

Remarks.—

State as shortly as possible in view of the facts any peculiarities noticed during drilling, such as bumping, very light or otherwise, movement or working of face or roof, sticking of drills on account of gas pressure closing holes; if striking 'rashings' increases flow of gas, or otherwise; any other remarks, etc.

His reports are turned in weekly, made up as here shown:

BOREHOLE REPORT

NO. 1 EAST MINE

Week ending May 28th, 1927 FIREBOSS .. Jas .. Maltman

Date	Place	Holes	Length	Gas	First	Tempe	rature	Remarks
Date	1 lace	110103	Dength	Jas	flow	Place	Hole	Temarks
23rd	2 off 19	1	19	Strong	12	54°	50°	Explosive gas
25th	X-cut off							
	2 R. 2 Left	1	18	66	11	54°	50°	66 66
25th	1 R. off							
	2 R. 2 Left	1	19	6.6	8	46°	42°	66 66
26th	1 R. off 30,			!				
	10 E.	1	20	"	12	44°	42°	66 66
26th	2 R off 30,							
	10 E.	1	18	Strong	10	44°	40°	Explosive gas
27th	3 Left X-cut	1	19	None		48°	52°	
27th	4 Left	1	18	Light	16	48°	54°	1" cap.

Temperatures have only recently been taken, following advice from Mr. Rice.

It will be noticed that there is a drop in temperature in the bore-holes when explosive gas is reported. This indicates that the gas met is under pressure in the coal. The writer believes that the drill only drains the gas from the surface exposed by the drill and not from the surrounding area.

On two occasions there have been outbursts from places that had been drilled a few hours previously.

One cannot say positively that the holes do no good, as it is impossible to tell whether there would have been more blowouts had the drilling not been done. The holes, however, have a good effect on the workmen, as most of them feel more at ease when their place has one or two drill holes ahead.

Many of the holes give off a considerable amount of gas, which oozes from the holes without force, sometimes for days, or until the hole is worked off. In one case recently two holes were drilled in a place that had been driven about 20 ft. beyond a point of a previous outburst. It was reported that gas was blowing very strongly from the holes. The place was working only on the afternoon shift. The next day it was carefully

examined and it was found that gas was not only issuing from the hole, but was coming from all parts of the face. When a flame safety lamp was held within a foot of any part of the face it was immediately extinguished. A shovel-full of coal was taken from the right side corner of the place, carried back on the roadway into the main air current, and a safety lamp set on the coal; it was extinguished. A similar experiment was tried with coal taken from the left side corner. Whilst the lamp was not extinguished, the gas flame extended to the top of the gauze, filling the lamp with flame.

The following night two more holes were drilled in the same place, but did not give off so much gas. The place continued giving off gas as stated for two days.

The writer is of the opinion that advance drill holes are of little use in preventing outbursts.

The experiences of others are very similar. Mr. Roblings, in his paper, states: "My experience has been that boreholes are useless under conditions such as those existing in the Pumpquart seam". As mentioned in the reply to the discussion on his previous paper, boreholes released only the gas from the coal bored out, and in no way reduced the pressure throughout the seam. This opinion is upheld by M. Ghyssen, who is quoted rather fully: "Messrs. Schorn, Watteyne and Maquet during the period 1885 – 1895 carried out a number of experiments in collieries containing large quantities of methane. The borings in the coal in the face gave pressures which bore no relationship to the fire-damp content of the seam. Thus, in one of the most dangerous seams of the L'Agrappe colliery, the pressure in no case exceeded one atmosphere. M. Maquet at the Fontaine L'Eveque colliery found boreholes very inefficient".

Mr. F. C. Cornet, in his article on the Belgian outbursts and entitled "Precautions taken to guard men from Seam Outbursts" (*Coal Age*, March 8th, 1923), states: "It became the general practice to drill the face with boreholes so as to drain off the gas contained in the seam. This practice long ago was found to be utterly ineffective; a worthless and illusory precaution. It was shown repeatedly by experience that faces well drilled and supposedly well drained of gas would blow out without warning other than that of the underground thunder

and crushing of timbers to which the men had been accustomed in seams at a higher geological level, including those in which no gas is ever found".

There is on record a case at Cassidy, which states: "In order to test the efficiency of exploratory holes, a place was chosen that was giving signs of an outburst. This place on September 29th, 1922, had three exploratory holes 20 ft. deep. With the exception of a 1/4-inch cap of gas coming from the flank hole on the rib side, not a trace of methane could be found. The place was stopped and remained idle until October 3rd, or more than four days. In the meantime 18 additional holes, 2 inches in diameter and from 20 to 24 feet deep, were drilled in an effort to tap the gas. With the exception of very slight gas caps at four of the holes no gas was tapped. A test with the Burrell gas detector at the highest point in the roof showed 1.4 per cent methane. The place was allowed to work on October 3rd, and after about 5 hours work, when about 10 tons of coal had been mined, an outburst occurred which was unusually violent displacing 250 tons of coal and knocking out nine sets of timber. An enormous quantity of gas was liberated for the next 36 hours. On return to the coal face it was found that from 3 to 6 ft. of three of the centre holes were in perfect order".

Investigation of Drill-hole Gas

An interesting article appears in the German publication *Gluck Auf* of date October 30th, 1926, entitled "The Investigation of Drill-hole Gas as means to foresee Gas Outbursts in Mines", by Mines Inspectors C. Kinderman, Waldenburg, and Diplom, and Mine Engineer L. Tolksdorf Molke, in which they describe a special apparatus for taking samples of gas from the inner end of boreholes. The apparatus is shown in Figures 62 and 63.

The apparatus consists of a telescopic casing with a total length of 1.27 meters, or about 50 inches, and a diameter of 1.6 inches. There is an expanded portion, c, which is a rubber hose fastened to the inner part of the casing. This hose is expanded against the side of the hole by means of an air pump,

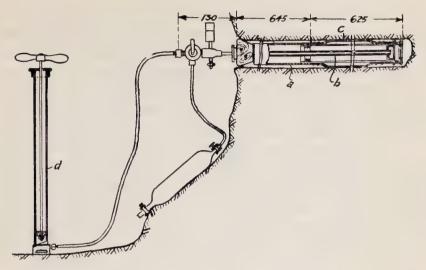


Figure 62.

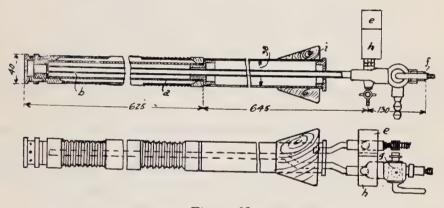


Figure 63.

Apparatus for taking samples of gas from the inner end of boreholes.

d. It will be noted that in the lower figure there are two places for connecting the air pump to the apparatus. One of these is for compressing the air in the case, which air, passing through small holes, expands the rubber hose against the sides of the borehole. The wood block, i, at the mouth of the hole through which the casing passes is merely for centring the casing in the hole, or, in case of the hole directed upwards, to wedge it in place until the gasket has been expanded against the side of the hole. Having forced the gasket against the side of the hole, the air pump is disconnected from the tap used for compressing the air and is connected to another tap, f. The pump then serves as a suction pump, first drawing the air and gas out of the

inner part of the casing through the tube which passes through the diaphragms to the inner end. Having drawn off the air, which in some cases, apparently, may be blown out by the gas behind it, the three-way stop-cock is closed and the connection is made to a gas-sampling flask which has a valve at each end. When it is necessary, gas may be drawn off by the suction of the pump and then forced into the gas-sampling container.

When the gas analysis shows a content of more than 30 per cent of the carbon dioxide, more severe limitations for blasting are made. When it reaches 40 per cent special precautions are taken. The statement is made that when there is 84 per cent of carbon dioxide, an outburst can be counted upon with certainty. It is stated that gas pressure from drill holes is practically unknown, on account of the porosity of the coal.

Indications and Warnings of Outbursts

After observing the characteristics of many outbursts, it has been found that there were several noticeable features attaching to them, which, if carefully checked, might be valuable as a forewarning of danger.

One of the main characteristics is the hardening of the coal face a few days before an outburst. This does not always happen, but if a place, working in a gaseous zone and giving off gas freely, suddenly tightens up and at the same time stops giving off gas, an outburst is usually looked for.

Then there is the cracking, splintering, and flickering of the coal; a bumping and general uneasiness of the roof over large areas; and, just a little before the outburst, there is rapid tapping, with sounds very similar to what is heard from the working of a percussive boring machine. This is followed by loud thundering noises. When the tapping starts it is time for the men to run.

It is said of the Belgian outbursts, "That of the hundreds of outbursts concerning which survivors have been able to give detailed and reliable evidence, there is no record of a single one having failed to announce its imminence in the same way".

The writer's experiences are similar, and it is by taking advantage of these warnings, constantly admonishing the men to be on the alert, and withdrawing from the face at the least sign of altered conditions, movements, bumping, or other warnings, that the outbursts owe their freedom from fatalities. In these mines, we have never had a man injured by outbursts, but some of the escapes have been very narrow. Some men take chances, especially if their place has been hard and suddenly loosens. Any man found taking undue risks is immediately removed from the outburst district, although there has not been much trouble in this regard.

The hard, tough coal met preceding outbursts varies in thickness from one to several feet. It does not always spread full across the working face, but may be only in parts. The miners say of it, "This looks like a big knot or boulder in the seam". The bigger or thicker it is, the greater is the force from the outburst on its removal.

On one occasion a place met one of these knots in the centre of a 14-ft.-wide place. It extended from points 2 ft. from each rib across the middle of the place. The corners of the place kept quite soft, and they were cut or sheared forward about three feet ahead of the centre. Suddenly the whole face pushed forward about 3 feet, until the corners or face of the shearing was up against the timbers, filling the sheared space solid and pushing the hard knotty coal forward like a huge piston. There was no gas given off or noticed on this occasion, and no interruptions of operations; the miners just kept on working.

When digging or cutting through these hard portions, there is hardly any gas given off. This is very unusual in such a gaseous seam. It was thought that this tough coal contained very little gas, and that this caused it to be hard and tough. Reference to the analyses on page 377 will show that the hard knotty coal contained 280.8 cubic centimeters of gas per 100 g. of coal, and the outburst coal 259.8 cubic centimeters. No doubt the outburst coal lost a large quantity of its gas at the time of blow-out.

Mr. Yant, Supervising Engineer, says of the coal samples: "The main significance to be attached to them is the fact that large amounts of gas are occluded in these samples as compared with Pittsburgh coal, and that this gas is rather tenaciously held in the coal. However, when the particles are fractured, that portion of pores or occlusions affected liberate their gas with comparative ease".

The Shot Jarring System

The adoption of a proper method of firing shots in outburst areas is now considered to be the safest means of safeguarding the men against the dangers from these occurrences.

It had been noticed in areas where outbursts occur that a too vigorous working of the face produced blow-outs. After a great deal of experimentation the Belgian Department of Mines permitted the practice of firing shots in the solid coal to jar or shake the strata sufficiently to provoke an outburst. This practice has been in vogue for a number of years, and hundreds of outbursts have been wilfully provoked, and none has caused an injury or scratch to any one.

The 'jarring shot' method was adopted some 16 years ago in the Gard coalfield of France. With this method boreholes are only used for the placing of shots, and these are placed in the coal in sufficient numbers, and so located, that their operation will shake the whole face or group of faces operated, and are fired simultaneously or in series and in rapid succession. The firing is done electrically from a safe distance in the mine by two experienced shotfirers, after all other men have gone home. When the men arrive in the morning, they clean up the loose coal, set all necessary timbers, and drill new holes in the face. The miners are strictly forbidden to cut into the face with the pick.

In a single year in one seam in the Moliere shaft, at Bessages, (Gard, France) forty-four outbursts took place, all of which resulted from shattering shots fired at night. None occurred otherwise, and not one of them killed or injured anyone. In the whole Gard district, not a single man has been injured with outbursts since November 15th, 1912.

Not a single inflammation of gas or dust has been recorded as the result of a shattering shot.

Tests at Ponthenry, South Wales

Mr. Robling, in his recent paper on Outbursts, refers to the disasters at Ponthenry, South Wales, in which there was serious loss of life on three different occasions, the last serious accident causing the death of five men. He then goes on to describe the steps taken with a view to putting into practice some means of safeguarding the men in his colliery from further dangers, and says in part: "The question of using explosives had been considered for some time, but fears of serious ignition of the gas militated seriously against its adoption. In order to obtain full information as to the methods adopted on the Continent and the position as to ignition, it was arranged in conjunction with Mr. Charlton, H. M. Inspector of Mines, and the writer (Mr. Robling) to visit Belgium and the south of France. In our investigation, we found that explosives were freely used to bring about outbursts. In the Gard coalfield, 'ripping' shots have been adopted for the past twenty years, and have undoubtedly been the means of preventing considerable loss of life'.

Under the regulations governing these areas, no picks are permissible in suspected places, and it is explained that this method of 'ripping' shots has been adopted to release the coal, the aim being to prepare and fire sufficient holes to blast as much coal as can be filled out in one shift, whether an outburst is brought out or not. This process is repeated daily, if necessary, and anything from 4 to 20 holes will be bored and fired in a shift. As in Belgium, the volleys are fired from a safety chamber not less than 200 meters from the working place.

The seam here, as in Belgium, continually gives off large quantities of methane, and in the Creal mine the return air is

analysed weekly and the quantity of air varied to suit.

They failed to find evidence of any *ignition of methane* as the *result of volley firing*, and the engineers in both Belgium and France expressed every confidence as to the safety of this system of releasing coal subject to outbursts. The whole of the information obtained led them to conclude that this was a safe system, inasmuch as the coal was being attacked in the absence of workmen, so that in the event of outbursts being induced, no persons were exposed to their dangers.

After some further consideration, it was decided to test this method at Ponthenry. In view of the fact that this method called for simultaneous firing of several charges in the coal, contrary to the provisions of the Explosives Order, nothing could be done until permission had first been obtained from the Mines Department to carry out tests. This was

readily given, subject to all details being arranged with H. M. Inspector, Mr. Charlton. It was ultimately decided to use dynobel No.3 in quantities not less than those employed in Belgium, *viz.*, 400 grammes (about 14 oz.) in each hole, and that the holes should be not less than 5 ft. deep.

The Mines Department asked Messrs. Nobel, the explosives manufacturers, to give them the benefit of their great experience, and two explosives experts, Messrs. Hiscocks and Weir, were present when the first tests were made. Experts agree that it would require a flame of a temperature of 1,700° to 1,800° to ignite the highest explosive mixture with a time exposure of 0.01 second. Mr. Hiscocks informed them that the flame of dynobel No.3 gave a temperature of 1,590° with a duration of 0.00025 second, while an appreciable period elapses before the gas is released, to which must be added the lag of ignition of fire-damp and air as well as the high percentage of the former. It is therefore safe to assume that little need be feared from this cause.

There were also present, when the tests were made, Messrs. G. C. Kirk, Managing Director of the Colliery Company; W. I. Charlton, Divisional Inspector of Mines; E. Sims Rees, Senior Inspector of Mines; and T. Walden, Junior Inspector of Mines.

Test No. 1:

A telephone was placed about 300 yards away from the seat of the test, and the receiver taken off so as to make it possible for a person at the surface telephone to listen for the report. The shots were fired from the surface, and the person at the telephone reported when the shots had gone off. The party conducting the test then descended the mine, and on reaching close to the point of discharge found the heading full of gas. On proceeding to the place of test they found that an outburst had taken place in front of the heading, ejecting some 30 tons of coal. The charges had all been fired, but from the 6th to the 11th there had been no effect other than the blowing out of the hole.

The following table shows the distance apart and depth of hole, and the weight of charges, in the first test. The explosive used was dynobel No. 3.

Hole	Distance apart	Length	Charge
1	In upper side	4 ft. 3 in.	18 oz.
2	4 ft. 4 in.	5 ft. 0 in.	14 oz.
3	4 ft. 8 in.	4 ft. 5 in.	14 oz.
4	4 ft. 6 in.	4 ft. 4 in.	14 oz.
5	5 ft 1 in.	5 ft. 0 in.	14 oz.
6	4 ft. 0 in.	5 ft. 0 in.	14 oz.
7	5 ft. 4 in.	5 ft. 1 in.	14 Oz.
8	4 ft. 2 in.	5 ft. 0 in.	14 oz.
9	5 ft. 3 in.	4 ft. 10 in.	14 oz.
10	4 ft. 11 in.	5 ft. 0 in.	14 oz.
11	3 ft. 11 in.	4 ft. 9 in.	18 oz.

The portion blown out in front of the heading would appear to have been the only part which had gas adsorbed at such a pressure that it could be considered as unsafe. It produced evidence of the variation in the amount of gas adsorbed throughout the soft coal.

The results obtained fulfilled all expectations. This first test showed that firing heavy charges of explosives would produce outbursts where conditions were favourable for such results.

Test No. 2:

Nine holes were fired, each 5 feet deep, and charged with from 14 to 18 oz. of dynobel No. 3. No outburst was brought about.

After this test the place was cleaned of the loose coal. The face was driven about 8 ft., when a few sharp reports were heard on the right side. All handwork in this place was stopped and another test was made.

Six holes were placed, charged with from 14 to 18 oz. of dynobel No. 3, and charges fired simultaneously from the surface. An outburst was brought about.

After several more tests it was concluded that the results proved conclusively the efficacy of this method for producing outbursts and thus removing with safety the dangers arising in outburst zones.

The results fulfilled all expectations, and notwithstanding the failure of some of the volleys to produce outbursts, the confidence of all, both staff and workmen, employed in working the seam has been restored.

A review of the outbursts produced also shows that the chances of ignition are very remote, when one considers the statement of Mr. Hiscocks that the flame of dynobel No.3 lasts for only 0.00025 second. Samples of gas were taken after some of the outbursts and showed 93 per cent methane.

The ability to attack these danger zones with the minimum amount of risk to human life, enabling new work beyond being opened up with improved economic results, is in itself a reward for the inauguration of this method. It now passes from the testing stage to that part of the ordinary working of such parts of the seam as may be required, and Special Rules were established to govern the working of the Pumpquart seam, Ponthenry. They are as follows:

SPECIAL REGULATIONS IN REGARD TO OUTBURSTS OF FIRE-DAMP IN THE PUMPQUART SEAM

Coal Mines Act, 1911, Section 87. Mining Industry Act, 1920.

- 1. It shall be the duty of all workmen to inform the duly appointed Fireman immediately of any indications of the likelihood of an outburst of gas in their working places.
- 2. (a) The duly appointed Fireman and Overman shall look out for any such indications in the course of their inspections and shall investigate immediately any information given them by the workmen.

- (b) If the Fireman or Overman suspects the likelihood of an outburst of gas, it shall be his duty to withdraw the men, to fence off the entrances of the place or places where such outburst of gas is suspected, and to inform the Manager or Under-Manager immediately.
- 3. If the Manager or Under-Manager, after inspecting the suspected area, considers that an outburst of gas is likely to occur, no further work shall be done in the suspected area except in-so-far as may be necessary to carry out these Regulations and to ensure the safety of the persons employed.
- 4. Volleys of shots shall be fired in the suspected area, under the personal supervision and direction of the Manager, with a view to provoking the outburst of gas. No person shall be below ground when this is being done.
- 5. Each volley shall consist of four or more shot-holes bored not less than five feet in depth with a rotatory drill, each charged with 14 ounces or more of a permitted explosive; and the shots shall be fired electrically in series from the surface of the mine.
- 6. The direction, depth and charge of each shot-hole shall be determined by the Manager or Under-Manager, and, in order to trace any misfire that may occur, shall be accurately marked on a plan which shall be taken to the surface of the mine before the volley is fired, and the direction and position of each shot-hole shall also be marked upon the roof of the working place with chalk.
- 7. (a) Immediately before each volley is to be fired, a thorough examination of the place shall be made by the duly appointed Fireman or Shotfirer, and the volley shall not be fired unless and until he finds that the place is free from inflammable gas and that there is no obstruction to the free circulation of the air current.
- (b) A similar examination of the place shall be made after the volley has been fired, and if any danger in the preparation or firing of subsequent volleys is found, it shall be removed before such work is proceeded with.
- 8. If more than one group of shot-holes has been prepared and all such groups are not fired simultaneously, the order of firing shall be in the direction opposite to that of the air current.
- 9. After all the volleys have been fired, the mine shall be inspected by the duly appointed Fireman or superior Officials before any other person is admitted.
- 10. The Manager shall keep at the mine a plan showing accurately the places at which outbursts of gas have occurred. A Register corresponding with the plan shall also be kept, giving the date of each outburst and the chief features characterising it, including the approximate weight of coal displaced and projected by the firing of each volley.
- Note. These Regulations are in addition to and not in substitution for the provisions of the Coal Mines Act, 1911, or any other Regulations made thereunder.

Mines Department, January 13th, 1927.

CONDITIONS AT PONTHENRY AND COAL CREEK COMPARED

At a Meeting of the South Wales Institute of Engineers held in April of the present year, Mr. Roblings, commenting on the discussion of his paper, made the following very significant statement:

"It appears from Messrs. Rice and Caufield's remarks that the conditions leading up to outbursts, except some of the signs immediately preceding, differ materially from those met with at Ponthenry.

"The system of blasting has been carried on successfully in the Gard for many years with no evidence of ignition, and it is reasonable to assume that similar success could be obtained in the Crowsnest mines, and from the description given of the conditions in the mines of those areas it appears that the dangers in the British Columbia mines are no greater; those in Belgium are classed as the most dangerous, while in the Gard the volume of air circulating has to be varied so that the percentage of gas does not exceed a certain definite amount, which is about equal to that permitted in Coal Creek mines. The coaldust dangers are similar and have to be combated prior to blasting.

"The geological structure would tend to lead us to assume that it had some effect in causing both bumps and outbursts in Coal Creek, and in that matter I referred in my paper (p.471) to the prevailing opinion in the south of France (viii) that the outbursts, being more frequent under the higher ground than under valleys, were due to heavier weight of the strata. Assuming this to be so, the conditions in Crowsnest, where there is a considerable cover of from 2,000 ft. upwards of strata with loose ends forming the sides of the valleys, and the coal mined directly from the side of the rising ground, would accentuate this weight; but these considerations do not fit either in Belgium or with us, and, as in all districts where outbursts are prevalent, they are confined to possibly one or two seams out of many, and thus leave the final solution of the problem still somewhat obscure. What is, however, of greater importance, is that a means has been found to anticipate them and provoke them in the absence of men within the danger zones. It is, however, important to exercise a wise discretion in the selection of the men necessary to work the places approaching suspicious areas".

THE CASSIDY OUTBURST

Mr. Rice, in his Report on Cassidy outbursts, made recommendations as follows:

- "1. To give a good trial to the plan of advance boreholes proposed by the Department of Mines, keeping a careful record of the results.
- If the advance bore-holes are found to miss the pockets, with consequent danger to men in subsequent mining, I suggest you take it up with the Chief Inspector of Mines and ask permission to blast when all the men are out of the mine, except shot-fires, of course using the utmost precautions and using rock dust or shale dust or flue dust immediately in front of the holes. It is assumed, of course, that a permissible explosive and clay tamping would be employed. Shot firing when all the men are out of the mine, the Department of Mines considers, would be in violation of the present Coal Mines Regulation Act, and the further danger from the accumulation of methane between the time of charging the holes and of blasting from the surface. For example, a blow-out might occur, and if the charge was pushed out with the coal en masse, the charge would practically be fired in the presence of methane, and no explosive under such conditions, even permissible explosives, would be safe to fire under such circumstances.
- "3. In the event of the failure of both the previous methods, I would propose long bore-holes of about 30 feet in length running close to either rib and near the top, and charging the inner end of these with two pounds of permissible explosive, using 6 feet

of stemming dust shelves at the mouth and firing from the surface, with the object of so shaking the ground that any point of gas under high pressure in the vicinity would be liberated. Such holes not being able to blow out the coal, and if a quick permissible is used will not blow out the tamping. so that the energy of the explosive would be expended in cracking the surrounding ground. Experiments, would, of course, have to be made to determine how close such explosives would have to be placed to the roof without damaging it. In the blow-outs inspected by the undersigned the gas seemed to blow out along the top of the coal. This may result that when expansion begins the gas draining from the crumbly coal in the pocket permits it to settle and leaves a space adjacent to the slickensided roof".

The British Columbia Department of Mines replied in effect that it was dangerous to fire shots in coal in the blow-out area where large volumes of gas were given off, and that it could not entertain the project of firing shots from the surface, as that was in violation of the Mines Regulation Act; further, had it not been in violation, the plan would not be approved as there would be danger of explosions or fires resulting which, in turn, might lead to the loss of life and property.

INTRODUCTORY REMARKS AND DISCUSSION

THE CHAIRMAN (MR. J. A. H. CHURCH): I am going to call on Mr. Shanks, General Manager of the Brazeau Collieries, Ltd., to introduce this paper.

MR. Shanks (Canada): This paper is not one that lends itself to discussion, because, apart from one or two details, it is descriptive and not controversial. I will, therefore, merely touch on certain points in connection with the coal industry of western Canada that may seem strange to our visitors from overseas.

In the first place, I may say that in my opinion, based on twenty years' experience in British Columbia and Alberta, the coal-fields of western Canada are at the present time overdeveloped. By that I mean that many more mines have been developed than can find markets for their produce. The Government is considering bringing in legislation to stop the development of new coal-fields, or any further development, until new markets are created.

I would like to point out to you—to give you an idea of the extent of the coal-fields of this Province (Alberta)—that a line drawn almost from the Arctic circle right down parallel with the eastern slopes of the Rocky mountains to the International boundary, crosses one huge, almost unbroken coalfield. Most of you would notice in passing through the coalfields here, and especially in the Crowsnest Pass district, a great many things which you could not understand. I had a similar experience when I was in Belgium in 1905. I saw some things I could not understand. I saw women pushing empty cars on one side of the pit-head while there was machinery on the other side hauling the loaded cars. They explained to me that labour was cheap and they only put in machinery when it was necessary; and for them, their great problem was to save coal, not labour. You who come from Great Britain, where everything of value is taken out of the coal, may wonder why we in Canada are not more up-to-date in our coking methods—why we do not recover the by-products. This, I should explain, is because at the present time there is no market for by-products in western Canada. Our land is so fertile that it does not require anything to further enrich it; also, natural gas and petroleum are cheap. As a result, it would not be economic to recover the by-products from coking our coal at the present time. The day will come, no doubt, when we will be taking everything of value out of the coal that it is possible to take out of it, but, because of lack of markets, it is not profitable to do so today.

The Chairman, I know, would like to start a discussion on gas outbursts and bumps, but I am not going to encourage him, because it might develop into a fight. Gas outbursts and bumps are common to the coal-fields both of Great Britain and of this country, but the physical conditions here are entirely different from those in Great Britain. I have had experience in Great Britain both as Manager and Engineer,

and I know we have physical difficulties and problems here which are not encountered in Great Britain. Our coal seams in the mountains are all more or less pitching, and in most of the fields we get into great cover very rapidly.

Those of you who have read this paper will notice that under 2,400 feet of cover we have considerable difficulty in connection with the mining of bituminous steam-coal because of the great roof weight. Now at 2,400 feet in the Old Country you have difficulties, but not to the same extent. Your coalseams are more or less flat, the roof strata have not been subjected to the same thrust pressures. You get 'caving' of the local roof, and subsequent upper strata interlocking.

In western Canada, with the same height or depth of overburden, we have greater difficulties to contend with. Here, the 2,400 feet of cover is generally a mountain, rudely cone shaped, with the shoulders cut away by ravines. Consequently we have great difficulty in controlling this 'dead weight'. I want you to appreciate these facts when you read this paper.

MR. NORMAN FRASER (Canada): I am fairly conversant with the domestic-coal situation in the Province of Alberta. When Mr. Shanks was speaking about the development of mines on the different railways, he was, of course, referring altogether to steam coal. We really have the same problems with domestic coal, but we would not have them to the same extent if our transportation difficulties could be overcome.

There are several ways of overcoming this, and one is for our railways to handle our coal at a lower rate. As being a matter of National importance and concern, it might be maintained that the government should bonus the railways to carry our coal to eastern Canada. There is argument for and against that, and it is a matter that deserves very serious consideration. We have a trans-continental railway here that has been built with the people's money and it is not altogether paying its way. If it could haul our coal at anything approaching cost, I think this would go a long way to help out the railway and the country as well.

It is said that every 500 tons of coal we can produce in the west will support a miner's family, and you can understand the great development that would follow in western

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Canada if we could market our coal. At the same time, it would keep in Canada the money which is at present going over to the United States. It is a matter that deserves serious consideration from an economic standpoint. I am a mining engineer and I do not profess to know a lot about economics; but surely there are those who can find the proper solution to this problem and tell us whether it would not pay us better in the end to lose a little money in the first instance and gain it in another way.

COAL WASHING AT THE LINSI MINE, KAILAN MINING ADMINISTRATION, NORTH CHINA

A COMBINATION OF THE BAUM, THE DRAPER, AND THE FROTH FLOTATION SYSTEMS

By Alex. Docquier,* L. Bataille, and R.Beetlestone

(Communicated by MAJOR W. S. NATHAN,**

Member, Inst. M. & M.)

(Sydney, N.S., Meeting, September 9th, 1927)

Introduction

The separation of coal from ash where the difference in specific gravity is considerable is an easy operation, and most of the well known washing plants give equally good results, differing in economic and other details according to circumstances.

When, on the other hand, a considerable proportion of the coal and ash is intimately mixed and the specific gravity of this mixture approaches that of good coal, the operation becomes much more complicated. This was the problem that the Kailan Mining Administration had to solve when it was found, some few years ago, that a considerable market could be opened up for its coal if it could be made suitable for coke manufacture.

In addition to lump coal, the mines put out a large quantity of dust coal with a high ash content, of which a considerable proportion is an intimate mixture of the two. None of the ordinary methods of coal washing were capable of removing the ash or coal-ash mixture efficiently and economically.

In the first place, it was found to be absolutely necessary to size the coal before washing it in order that the subsequent washing should be efficient.

^{*}Engineer-in-Chief, Linsi Mine.

^{**}Late Agent and General Manager, Linsi Mine.

The mines possessed two washing plants, No. 1 and No. 2, on the Baum system without sizing arrangements, which were not sufficiently effective for the required purpose. It was necessary, therefore, to instal another plant to deal with the inferior coals, first of all to separate the coarse from the fine portions and then to wash them. The plant finally decided upon was a combination of a Draper plant and a flotation plant.

The coal was sized by means described in the Report, and the coarser grained coal was then washed in the Draper plant; this plant cannot, however, deal with the fine portions owing to the clogging of the water in the tubes. The flotation plant, on the other hand, is only suitable for washing the finest portions. A combination of the two systems, however, has been found to give good results, and allows of much elasticity both as regards the nature of coal treated and the production of a coal of any desired characteristics.

A description of the plants, of the methods of working them and of the results obtained, are given in the following pages.

We have to thank Mr. F. S. Sinnatt for his valuable advice and assistance in connection with the revision of the report for publication.

CHARACTERISTICS OF THE COALS

The coals differ largely in their characteristics as compared with European coals. To show this more clearly, samples have been taken with extreme care from the various beds of the workable seams. Sections have been made showing the composition of the seams with their respective ash percentages. Details are given in Figure 1.

A consideration of the sections will show that there exist, even in a single seam, many varieties of coal which differ in ash percentage from 6 per cent to 50 per cent in the case of band coal.

It may at first be thought that the action of washing merely separates the parts of the seam of low-ash from those of high-ash content. Samples have, however, been taken of the various grades, and washing tests on these show that even in the parts of the seam with high-ash content it is possible to recover a certain percentage of low-ash coal. The following are examples of the results obtained:

- COMPOSITION OF SEAMS

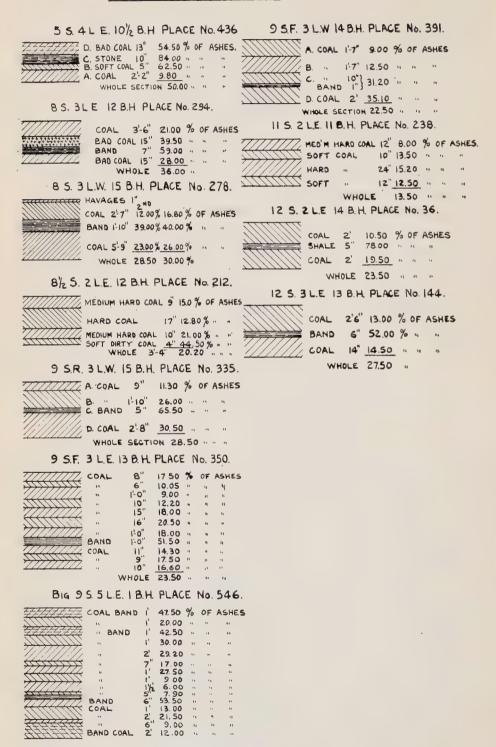


Figure 1.—Composition of coal seams, Linsi mine.

9 seam roof, 3 level, W. 15 B. H. Place No. 335, Part A (Original coal contains 10.3 per cent ash)

Р	X	A	B/A
% of weight of each layer	% of ash of each layer	% of weight, cumulative	% of ash, average cumulative
6.42	9.0	6.42	9.0
6.95	9.5	13.37	9.2
7.28	9.6	20.65	9.4
7.47	9.7	28.12	9.4
7.47	9.7	35.59	9.5
7.64	9.8	43.23	9.6
7.81	9.9	51.04	9.6
7.81	10.0	58.85	9.7
7.81	10.2	66.66	9.7
7.81	10.3	74.47	9.8
8.17	10.4	82.64	9.8
8.33	10.5	90.97	9.9
9.03	15.2	100.00	10.4

9 seam roof, 3 level, W. 15 B.H. Place No. 335 Part B (Original coal contains 20.50 per cent ash)

P	X	A	B/A
% of weight of each layer	% of ash of each layer	% of weight, cumulative	% of ash, average cumulative
6.48@	9.5	6.48	9.5
6.97 "	10.5	13.45	10.0
6.97 "	10.5	20.42	10.1
7.13 "	10.7	27.55	10.3
7.28 "	11.0	34.83	10.4
7.47 "	12.5	42.30	10.8
7.47 "	12.5	49.77	11.1
7.47 "	14.0	57.24	11.4
7.78 "	15.5	65.02	11.9
8.08 "	15.7	73.10	12.3
8.43 "	17.5	81.53	12.9
8.92 "	20.0	90.45	13.6
9.55 "	54.0	100.00	17.4

8 seam, 3 level, W. 15 B. H. Place No. 278 Part A (Original coal contains 12.00 per cent ash)

P	X	A	B/A
% of weight of each layer	% of ash of each layer	% of weight, cumulative	% of ash, average cumulative
6.24 @	5.9	6.24	5.9
6.38 "	6.0	12.62	5.9
6.38 "	6.0	19.00	6.0
6.70 "	6.0	25.70	6.0
6.85 "	6.0	32.55	6.0
6.85 "	7.2	39.40	6.2
7.02 "	7.3	46.42	6.3
7.60 "	7.5	54.02	6.5
7.60 "	7.5	61.62	6.6
7.77 "	7.5	69.39	6.7
8.07 "	7.8	77.46	6.8
9.14 "	8.5	86.60	7.0
13.40 "	22.5	100.00	9.1

8 seam, 3 level, W. 15 B. H. Place No. 278 Part B (Original coal contains 36.00 per cent ash)

P	X	A	B/A
% of weight of each layer	% of ash of each layer	% of weight, cumulative	% of ash, average cumulative
4.97@	9.0	4.97	9.0
6.07 "	12.3	11.04	10.8
6.63 "	12.5	17.67	11.4
7.32 "	13.0	24.99	11.9
7.32 "	15.0	32.31	12.6
7.74 "	18.0	40.05	13.6
7.74 "	20.5	47.79	14.7
7.74 "	27.5	55.53	16.5
8.00 "	32.5	63.53	18.5
8.00 "	36.0	71.53	20.5
8.29 "	47.5	79.82	23.3
8.58 "	47.0	88.40	25.6
11.60 "	53.5	100.00	28.8

For fuller explanations of the above Tables, see page 412.

METHOD OF TESTING THE COALS

The products that may be expected by treating the raw coals in the washing plants are ascertained in the first instance by testing samples in a special hand washing machine at the laboratory. From these tests, washability curves are made, showing the products actually contained in the coals.

The small hand testing machine installed was made in the workshops (see Figure 2), and consists of a 12-inch diameter tube shaped in the form of a U. In one leg of the U a plunger (3) is placed, operated by handles; in the other leg, one half length of which is moveable, above the joint and about midway in the leg, a grid (2) is fixed supporting a perforated plate with holes of 30-mesh per inch, which forms the bed to receive the coal to be tested.

At the bottom of the U, a plug (8) is placed for the evacuation of the fine coal or slurry, which may be drawn through the perforated plate during the washing process.

About 50 lb. of the coal to be tested is placed on the grid, and the machine almost filled with water. Jigging is carried out for one hour, during which period the coal settles down in the tube on top of the perforated plate in layers according to its specific gravity.

By means of the side screw (5), the portion of the tube containing the coal is moved over the table, where a moveable blank plate (6) is situated, slightly less in diameter than the tube. The screws (7) holding the grid are removed and the blank plate elevated, lifting the grid and its load of coal up the tube to the spout level.

When the coal reaches the spout level, the inches on the finger attached to the elevating screw are read, and it is then pushed upward one inch, the coal levelled, taken off, dried, weighed, and analysed.

It is then pushed up another inch, and the coal again levelled, taken off, dried, weighed, and analysed as before; and so on, until the whole of the coal has been dealt with.

The slurry is then tapped off, dried, weighed, and analysed.

From the data thus obtained in the laboratory, the curves of washability are determined. They show clearly what is in the coal, and may be termed *the theoretical result*.

Typical results are given in Tables 6 to 12, and washability curves in Figures 6 to 12, at the end of this paper.

The following is an explanation of the figures in the different columns of the Tables:

- Col. 1.—This contains the reference number of each layer of coal as taken from the testing apparatus.
- Col. 2.—Gives the absolute weight of each of these layers.
- Col. P.—Gives the percentage of weight of each layer, slurry omitted.
- Col. A.—Gives the cumulative weight percentage.
- Col. X.—Gives the percentage of ash of each layer.
- Col. PX.—Gives the percentage of weight multiplied by the percentage of ash of each layer.
- Col. B.—Gives the cumulative weight of column PX.
- Col. B/A.—Gives the characteristic of the washed coal averaged. For example, in Table 6 take layer No. 10: In the column B/A is the figure 11.7. This figure represents the average percentage of ash of the whole of the layers from 1 to 10 inclusive.
- Col. C.—Gives the cumulative weight percentage in the opposite way to Col. A. For example, in Table 6 take layer No. 10: In the column C is the figure 32.37. This figure represents in weight percentage the sum of layers 10, 11, 12, and 13.
- Col. D.—This column D is to Col. C. what column B is to Col. A.
- Col. D/C.—This corresponds for the ash percentage in the stones in a similar way to Col. B/A.

For example, in Table 6 take layer No. 10: In column B/C is the figure 44.1. This figure represents the average percentage of ash in the beds or ayers 10, 11, 12, and 13.

In the washing charts, in the columns in the small table dividing the products in ten parts:

%—Indicates the weight percentage.

Coal—Corresponds to Col B/A in the corresponding Table, giving the average percentage of ash of the cumulated coal.

Ash—Gives the percentage of ash of each bed.

It will be seen from the washing curves that the raw coal is made up of:

- (a) A percentage of fairly pure coal.
- (b) A percentage of coal and intermixed band coal.
- (c) A percentage of shale or refuse.

The regularity of the curves indicates that absolute separation of the products is impossible. The difficulty of obtaining efficient separation is also increased by reason of the very small difference in the specific gravity of the coal and of the coal with intermixed band coal on the one hand; and the dirtier intermixed band coal and shale on the other hand.

The raw coal treated at the Linsi washing plants may be roughly classified into slack and nuts. The size of the slack is from 0 to 30 or 35 mm., of the nuts from 30 mm. or 35 mm. to 70 mm. The following table shows the normal ash content of these varieties:

		per cent ash
(a)	Linsi special slack	. 20 to 25
	Chaokochwang special slack	17 to 21
	Tangchiachwang special slack	
(b)	Linsi 1 slack	
	Linsi 2 slack	. 23 to 27
(c)	Linsi nuts	. 27 to 31
	Chaokochwang nuts	. 28 to 33
	Tangchiachwang nuts	. 25 to 30

DESCRIPTION OF No. 1 AND No. 2 WASHING PLANTS, WITH RESULTS OBTAINED

Description of Plant

For the flow sheet of the plant see Figure 3.

The coal is unloaded from cars (1) into a bunker (2) below rail level. The bunker is conveniently bevelled to admit of a continuous and regular discharge of the coal by gravity into an elevator (3). The elevator is about 100 feet long, and a shoot (4) is fixed at the top to pass the coal into the washing box (5), where it is washed.

The heavy and the light refuse is evacuated from the washing box by an elevator (6) and is finally passed over a shoot and deposited in bunkers feeding railway cars for sending the product to the dump.

The coal, now cleared of its refuse and high-ash coal, enters over the second washing box (7), where it may be divided into first- and second-quality coal.

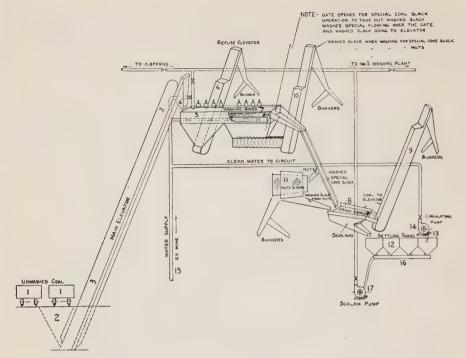


Figure 3.—Flow sheet, No. 1 and No. 2 washing plants.

After going through this second washing box, the first-quality coal recovered is passed over a drain plate (8) to remove as much as possible of superfluous water, and from here it is elevated to the elevator (9) for distribution to bunkers feeding railway cars.

The second-quality coal is evacuated from the washing box in a similar manner to the refuse from the first box, and this product also is discharged into an elevator (10) for delivery into bunkers feeding railway cars.

In the operation for washed nuts, the coal product is by-passed to a revolving screen (11), where the slack is taken out in order to obtain pure nuts of first-class appearance.

The bunkers into which the washed products are delivered are fitted with tubes containing elongated slot holes — the opening gate is likewise perforated — for further drainage of superfluous water. The tubes and gate are connected to a drainage system for taking away the water.

This drainage water contains an amount of recoverable fine coal and is led to a series of outside settling tanks, where the flow of the water is reduced until a point is reached when the still water settles out its solid content. When one series of reservoirs is filled with solids, the drainage water is led to another series so as to allow the first series to be left for a few days for further drying-out of as much moisture as possible. The solids or slurries are afterwards recovered and used as boiler or brickworks fuel.

Reverting to the question of the water drained from the first-quality coal after leaving the washing box. This water, considerable in quantity, and carrying with it a proportion of fine coal, is led to a series of inside settling tanks (12) shaped like inverted pyramids.

The water, in passing from one to another, is retarded in its flow, permitting the solids to sink to the bottom. The water becomes clearer in its travel, and, on arriving at the last tank of the series, is suitable for re-use. A suction pipe (13) is therefore introduced, connected to the main circulating water pump (14). Make-up water (15), to compensate for that which the coal has picked up in moisture content, is also connected to the same pipe.

At the bottom of each tank a specially constructed valve allows an easy discharge of the settled slurry when it reaches a level which influences the clearness of the overflow water.

A common drain (16) is provided in connection with the tanks, and the thickened slurry withdrawn from time to time is conducted by this drain to a collecting reservoir feeding a slurry pump (17).

From the pump the slurry may be:

- (a) Sent back to the circuit, meeting the raw coal at the entrance to the washing box.
- (b) Drained off separately.
- (c) Added to the drainage system to outside settling tanks.

It is generally arranged for the slurry to re-enter the circuit.

It may be explained that these slurries cannot be treated in No. 1 and No. 2 washing plants. When, therefore, the first-quality coal becomes too high in ash owing to the presence of too great a proportion of untreatable slurry, it is necessary to by-pass the slurry until the ash content of the whole is diminished so as to admit of further quantities.

Results Obtained

The following are the results and yields obtained at No. 1 and No. 2 washing plants:

1. From Linsi special slack

	Raw coal, ash	Product	Recovery	Ash
	22 to 26% $\begin{cases} 1 \text{st q} \\ 2 \text{nd} \\ 3 \text{rd} \end{cases}$	quality coaldo. do. do.	per cent . 22.00 . 43.00 . 35.00	per cent 11.00 17.50 43.00
2.	From Linsi 1 slack			
	$ \begin{array}{c} 26 \text{ to } 28\% \dots \\ 2\text{nd} \\ 3\text{rd} \end{array} $	quality coaldo. do. do.	. 22.00 . 43.00 . 35.00	11.40 19.30 48.00
3.	From Linsi 2 slack			
		quality coaldo. do. do. do.	. 23.00 . 42.00 . 35.00	11.60 18.50 43.00
4.	From T.C.C. and C.K.	C. special slack		
	(4)	quality coaldo. do. do. do.	. 40.00 . 42.00 . 18.00	10.60 17.00 40.00
5.	From nuts			
	28 to 33% {Pure Slack Sla	e nuts	. 20.00	18.20 15.00 42.00 58.00

The comparatively small recovery of nuts under (5) is due to the presence of slack in the raw nuts and to breakage in handling. This slack is screened out after washing and is good enough to go as second quality coal.

It may be noted from the washing charts, as also in the actual washing operations, that, as in the case of Linsi 1 and Linsi 2 slacks, the recovery of first-quality coal is low, with a correspondingly high ash percentage; these coals are therefore usually reserved for straight washing to produce a second-quality coal, giving results as follows:

From Linsi 1 slack

F

Ash 26 to 28%	2nd quality coal	Recovery per cent 65.00 19.00 16.00	
From Linsi 2 slack	2nd quality coal	70.00	16.10 38.00 58.00

Particulars of No. 1 and No. 2 washing plants are as follows:

Each plant contains:

One main motor 150 h.p., operating at 2,200 volts a.c.

running at using 480 r.p.m. using 15 amps.

One main circulating pump and motor

Motor 50 h.p., operating at 2,200 volts a.c.

running at 480 r.p.m. using 7 amps.

Pump, diameter of suction pipe 230 mm.

" "impeller 570 mm.
width " " 40 mm.
Diameter of delivery pipe 230 mm.

The pump is direct coupled, and is capable of giving 5 c.m. per minute at 6 metres head.

One slurry pump and motor

Motor 65 h.p., operating at running at running at using 10 amp.

Pump, diameter of suction pipe 150 mm.

" "impeller 375 mm.
width " " 40 mm.
diameter of delivery pipe 150 mm.

The pump is running at 12.50 r.p.m., is belt driven, and is capable of giving 2 to $2\frac{1}{2}$ c.m. of slurry per minute at 30 metres head.

Blower

Diameter	of	$suction\dots\\$		u	6					0		a		310	mm.
"	"	delivery			۰				0		4			300	mm.
Length of	bo	ody						 ٠	ø	4	e e	0	.1	,000	mm.
Diameter	of	body					0 1	 0	4		0	٠	.1	,000	mm.
"	"	rotor												800	mm.

There are 4 wings to the rotor, and the machine operates at 1.15 r.p.m.

The capacity of each plant varies according to the coal washed, as, for example, from 60 to 70 tons per hour for the production of washed special coke slack, to 80 tons and more per hour for the production of washed slack or nuts.

DESCRIPTION OF NO. 3 WASHING PLANT.

This plant was designed as an experimental plant to incorporate the *Draper system* and the *Froth flotation process* for washing the coals, the Wendell and Elmore systems of centrifugal drying for the 'grain' coal, and the Oliver and American filtering processes for dewatering the flotation product.

For the flow sheet of the plant, see Figure 4.

The Draper System

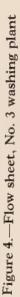
The installation takes the raw coal from two unloading tracks, and passes it to a masonry pit, fitted with a regulating gate for evenly feeding an elevator 100 feet in length. The coal is elevated over a shoot and, prior to washing, is treated in a series of revolving screens operating in water, for the purpose of obtaining as efficient sizing as possible.

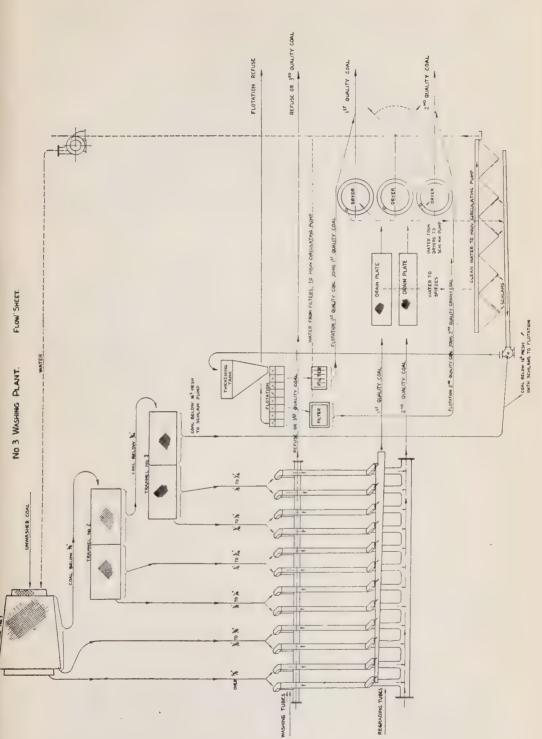
The sizes of holes in the screens may be modified, and screening results of different sizes will be found later in this paper, but for the moment the following will serve as a guide:

Over $\frac{5}{8}$ -inch From $\frac{5}{8}$ -in. to $\frac{3}{8}$ -in. " $\frac{3}{8}$ -in. to $\frac{1}{4}$ -in. " $\frac{1}{4}$ -in. to $\frac{3}{16}$ -in. " $\frac{3}{16}$ -in. to $\frac{1}{8}$ -in. " $\frac{1}{8}$ -in. to 16-mesh. Below 16-mesh.

The two largest sizes are taken from the screen by shoots, the next four sizes are evacuated from the screens by drag conveyors, and each of these six sizes of coal is then delivered to a pair of washing tubes.

As the sizes approach to, say, 1/16-in., it is apparent that, besides the mechanical difficulty of effective screening, a point is reached where the upward flow of water required to





float off the coal is so slight that the system is no longer practical. Below this size the coal is therefore sent to the flotation machine for treatment.

The washing in the Draper tube is explained in Messrs. Sinnatt's and Mitton's paper on "The Preparation of Coal for the Market" read at the first (1924) meeting of the Empire Congress in London.

After washing, the coal products may be taken separately over mechanically operated drainage conveyors for the elimination of as much superfluous water as possible. The coal is then conveyed to centrifugal dryers, descriptions of which are given later on in this paper.

The Froth Flotation Process

The flotation machine is of the mineral separation type. The present-day lack of proper understanding of the facts of surface tension makes it difficult to define the phenomena of flotation. The following features enter into the consideration:

- (a) Forces acting at the surface of a liquid, the resultant of which tends to prevent rupture of the surface.
- (b) The natural tendency of some substances to resist wetting.
- (c) The natural tendency of some substances to be wetted.
- (d) The natural tendency of some substances to take on to their surfaces a film of oil.
- (e) The natural tendency of some substances to resist taking on to their surfaces a film of oil.

Coals generally possess qualities (b) and (d); shales and refuse usually found with coal possess more or less the qualities (c) and (e).

Thus, by mixing a small percentage of oil with the feed coal, the coal readily assumes an oil film on its surface.

The agitation introduces a large quantity of air into the mixture of fine particles, oil, and acid. Then, when the pulp passes through the restricted outlet in the stirring compartment to the quiescent zone of its accompanying tanks, innumerable bubbles are formed in the latter, exercising a selective action

on the fines which results in the coal being floated off in the from of a froth. Meantime, the shaley matter, which has had its natural quality of sinking in water increased through the water having been rendered acid, quickly sinks, and the separation is thus effected.

It may be mentioned here that most recent American opinion considers that there are two classes of oil reagents, one of which is soluble in water and the other insoluble. The first has the property of producing the froth and thus increases the area of superficial tension; the second serves to film over the particles, and thus render them more floatable.

From this it would be theoretically possible to so calculate that the quantity of the froth-producing soluble oil should be proportional to the solids and the quantity of insoluble oil filming the particles be proportional to the total surface area of the floatable mineral.

Against this, however, is the fact that it is possible to have soluble and insoluble elements in a single oil, as well as the big difficulty, when dealing with coal, of maintaining uniformity in the proportion of solids in the feed.

The flotation machine installed is of the 8-box type. The first mixing chamber receives the coal to be treated and its proportion of water, to which is added the necessary percentages of oil and acid.

The mass is agitated in the chamber and forced through an aperture communicating with a tank. Part of the coal is floated off at the top of the tank as the washed product, and the remainder is drawn through a pipe connection at the base, controlled by a regulating valve, where it is sent to the second chamber, in which the agitation operation is repeated and part of the coal floated off as before; and so on through the series of eight boxes, the product from the last box being rejected as refuse.

The coal is collected from the eight boxes and divided into two qualities, the first three boxes yielding 1st quality coal, and the last five boxes combined a 2nd quality coal.

The refuse withdrawn from the last box is pumped to de-watering tanks for drying.

The reagents used per ton of material treated are:

Paraffin, 0.10 to 0.80 lb. per ton. Cresol, 0.30 to 0.60 lb. per ton.

The coal is passed from the flotation machine to filters for de-watering.

Results of Screening Tests

As the system is dependent on the efficiency of the screening prior to the material entering the washing tubes, details of the screens are given in Figure 5.

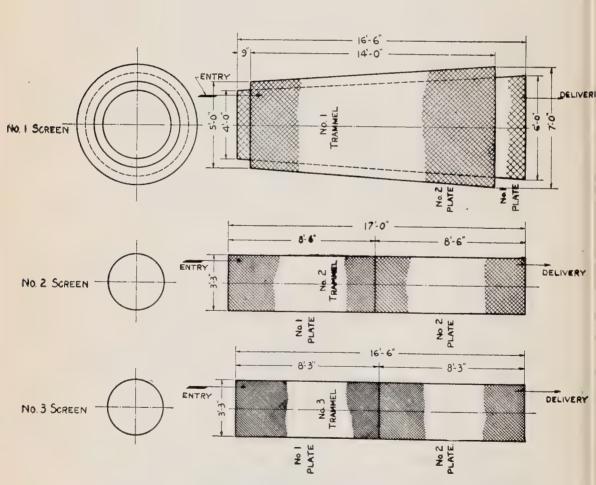


Figure 5.—Detail of screens, No. 3 washing plant.

According to the theory of falling particles of unequal specific gravity in still water, with a shale of specific gravity 2.3 and a coal of specific gravity 1.3, the ratio between the sizes at the time when the velocity-fall will be the same is 4.33. To obtain a separation by this means, therefore, the holes in the screen plates for the various grades of sizes must be kept within this limit and actually should be very considerably less. In practice on a large scale, it will be found that many other considerations are involved, such as the specific gravity of the water being no longer unity after a few moments of washing; the feeding of the raw coal in mass; the composition of the raw coal; the similarity of the specific gravities of pure coal and pure shale through the presence of intermediate band coals; as well as the fact that a variation in speed of falling occurs between the same particles passing through the same screen holes but of different shapes. For these reasons so large a ratio between the sizes as 4.33 cannot possibly be adhered to. In practice, it was not much greater than 2.

The following are details of the screening tests:

SCREENING TEST 'A'

145^T Linsi Special at 30^T per hour

Holes in 1st screen	.16	mm.	and	10	mm.
Holes in 2nd screen	. 8.5	mm.	"	4.8	mm.
Holes in 3rd screen	. 2.5	mm.	44	1.3	mm.

Size of coal	Hand screen results	Washing pl	ant results
Over 16 mm	19.68% 6.68 " 12.68 " 11.75 " 15.68 " 10.87 " 20.50 "	30.3 ^T 13.3 " 29.7 " 20.6 " 21.0 " 8.1 " 22.0 "	21.0% 9.2 " 20.5 " 14.2 " 14.5 " 5.6 " 15.2 "

TABLE 1 SCREENING TESTS

4	424 I	Docquiei	r, Batai	LLĖ AND	BEETLESTON	NE .	
	Av. speed at periphery	ft. per min. 220 260	160	160	220	160	160
13	Approximate Av. speed at tonnage per periphery sq. ft. p/h.	.155	.325	.24	.15	.29	.14
	Screening area	sq. ft. 258 267	98	85 55	258 267	98	85
	Length of screen	ft. in. 16 6 14 0	8 8	∞ ∞ ∞ ∞	16 6 14 0	9 8	8 8
SCREENING 1ESIS	Diam. of screen at delivery	ft. in. 6 0 7 0	ო ო ო ო	00 co	0 2	00 co	3 33
SCRI	Diam. of screen at entry	ft. in. 5 0	m m	00 00 00 00	4 c 0 0	n n	e e e
	Size of holes	mm. 16 10	8.5	1.3	11	9 6	1.3
	TEST 'A'	No. 1 trommel: No. 1 screen plate No. 2 do.	No. 2 trommel: No. 1 screen plate No. 2 do	No. 3 trommel: No. 1 screen plate No. 2 do	No. 1 trommel: No. 1 screen plate No. 2 do	No. 2 trommel: No. 1 screen plate No. 2 do	No. 3 trommel: No. 1 screen plate No. 2 do

Samples were taken from above washing plant products and hand re-screened, giving the following:

Size of coal		Bunkers No.							
		1	10	5	3	9	7	8	
Over " " " " Below	4.8 2.5 1.3	mm. mm. mm. mm. mm. mm. mm.	68.5 11.9 9.2 3.8 3.6 1.0 2.1	25.7 31.6 6.0 5.2 6.5 24.1	50.5 23.7 16.5 3.7 4.5	28.2 35.6 16.6 19.3	64.6 22.2 12.2	64.0	
	1.0		100.1	99.1	98.9	99.7	99.0	99.5	

SCREENING TEST 'B'

205[™] Linsi Special at 47[™] per hour

Holes in 1st screen	16 mm.	and	11	mm.
Holes in 2nd screen	9 mm.	ш	6	mm.
Holes in 3rd screen	3 mm.	"	1.3	mm.

Size of coal	Hand screen results	Washing plant results
Over 16 mm. 16-11 mm. 11- 9 mm. 9- 6 mm. 6- 3 mm. 3- 1.3 mm.	16.00% 7.00 " 4.00 " 15.00 " 13.50 " 19.00 "	17.4% 18.0 " 13.9 " 25.3 " 9.3 " 5.6 "
Below 1.3 mm.	26.00 "	10.5 "

Samples were taken of the above products and of hand re-screened products, and the following results were obtained:

Size of coal	16	16-11	11-9	9-6	6-3	3-1.3
Over 16 mm	74.2 8.2 1.6 4.4 1.6 1.6 8.8	56.0 3.0 21.5 5.5 4.5 10.5	37.0 29.0 15.0 12.5 8.0	55.0 11.0 14.3 18.7	73.5 20.0 6.5	59.5 38.5

Washing Results from No. 3 Washing Plant

The following are the results obtained from the combined plant:

1.	Linsi special slack	ξ			
	Raw coal, ash	Produ	ct	Recovery	Ash
	·			per cent	per cent
		1st quality	coal	31.00	11.3
	22 to 26%	2nd do.		40.00	16.5
	, ,	3rd do.		29.00	46.0
2.	Linsi 1 slack				
		1st quality	coal	25.00	11.7
	26 to 28%	2nd do.		41.00	17.0
	70	3rd do.	coal	34.00	47.0
3	T.C.C. and C.K.C	. special sl	ack		
0.				55.0	11.0
	16 to 20%	2nd do.		20.0	16.0
	20 20 20 /01 11 11 11	3rd do.	coal	25.0	48.0

Flotation Results

Tables 2, 3, and 4 give the results obtained from the flotation process alone operating on various slurries recovered from No. 1 and No. 2 washing plants.

Table 5 gives the result of the flotation process operating with No. 1 washing plant (Baum system), where the slurries were by-passed to the flotation plant.

FLOTATION RESULTS. WASHING PLANT No. 3 TABLE 2

	Oil used lb. per ton	C.A. P.	87 60 4400 394	94 7/0 507/0 384	76 20 5200 714	89 80 4300 628	87.70.3640.508	.410 0.571		010.64010.128	87.50.5300.070	91.50.5071.010	60.710.61610.585	92.80.6160.873	90 20 4650 632	85 90 3700 643	304 0.538		3780.698	.370 0.639		-
Is	otoT &		27 60	0/2 70	06 92	20 8	37.70	83.50.		186.010	37.50	11.50	30.70	92.80	200	25.90	68.60.	-	3.310	34.60	9.1	
	3rd quality coal	Tons	14 8 14 0 23 0 28 4 28 5		33.020.113.4 42.125.731.3						15.114.5 35.431.531.5		39.0 20.6 31.5						32 23.9 26.53 93.3 0.378 0.698	9 13.46 27.1 21.4 26.00 84.6 0.370 0.639	40.313.6759.122.726.2689.1	, , _ , _ , _ , _ , _ , _ , _ , _ , _
Washed coal	2nd quality coal	Tons % Ash	12 14 8 14 0	24 26 716 3	33.020.113.4	26.013.814.48	39.027.814.33	73.0 32.0 13.87		8 26.4 14.8	17 15.1 14.5		46.624.114.0	22.028.514.0	13.3 9.814.5	36.0 29.8 13.73			52 38.8 13.88 32	53 41.		Ш
	1st quality coal	Tons % Ash	36 44 4 11 5	48 53.311.6	163.9 18.4 49.9 30.4 10.8	91.048.511.3	140.2 17.98 46.5 35.3 10.3	227.9 18.26 56.0 24.5 10.1	TABLE 3	17 56.3 11.5	46 40.911.5	3 9.5 11.5	30.4 16.0 10.9	30.2 39.2 10.7	134.4 17.0 82.2 61.1 11.25	122.717.1323.018.710.20	224.0 17.60 64.0 28.5 11.20	TABLE 4	134.0 18.4 41.1 30.6 10.15	5 18.24 27.0 21.3 10.67	260.5 18.32 68.1 26.1 10.41 105	
al	Tons Ash		81 17.7	90 18.2	163.918.4	187.518.0	140.2 17.98	227.9 18.26	T,	30.2 15.7	112.4 19.2	31.5 18.9	189.2 17.2	77.017.0	134.4 17.0	122.717.13	224.017.60	TA	134.0 18.4	126.5 18.24	260.5 18.32	
Unwashed coal	Grade		Linsi special slack	33	×	"	**	ű		C.K.C. special slack	3	33	"	"	×	**	29		Linsi 1 slack	29		
	Operation No.		က	2	15	16	18	19		7	∞	6	13	14	17	20	22		32	37		
Tone	per						6.5			7.0	7.5	7.9	6.4	6.5	5.6	6.4	7.9		5.9			

Drying

After washing, the products are dried prior to bunkering, the 'grain' sizes being sent to centrifugal dryers, and the flotation product to filters for de-watering.

For the 'grain' coal, the principle of drying is the same in both the Elmore and Wendell systems, the machines differing only in mechanical detail.

The Wendell Dryer:

In the Wendell dryer, a conical-shaped container, some 12 feet in diameter, revolves at 250 r.p.m. in a horizontal plane. Inside of this a pair of distributing shoots revolve at a slightly different speed (say 1 per cent).

The outer wall of the container is of finely perforated metal through which the water from the wet material is ejected by centrifugal force. The water is discharged by a sluice round the machine, and the dried coal is discharged by means of a cam mechanism, which opens and closes the hinged discharge gates situated in the bottom of the container.

The bottom of the container is divided into 16 compartments, each provided with a hinged gate. The distributing shoots, revolving at the slightly higher speed, fill up the compartments with a charge of wet coal. The coal is then subjected to centrifugal action until the distributing shoot again overtakes the compartment to recharge, immediately prior to which the cam mechanism opens the gate to discharge the dry coal and closes it to receive a new charge.

The Elmore System:

In the Elmore system, the mechanical features are different. The coal is fed to the conical basket and is held against the screen plates by helical-shaped scrapers.

Results of Drying:

Prior to drying in the machines, the coal contains between 12 per cent to 20 per cent moisture, which is reduced in the machines to very little more than 8 per cent when the screen plates are new, but only to about 10 per cent as the holes in the plates become worn. When this stage is reached, the screen plates are renewed.

Arrangements are provided whereby the coals, after drying, are combined with the corresponding qualities coming from the flotation machine, and are then elevated to bunkers for loading into cars.

Filters:

The filters employed are of the Oliver and American types, in which the principle of de-watering is the same, but the mechanical details are different.

In the Oliver filter, a drum slowly revolves on a horizontal axis, with the lower portion submerged in a tank into which the flotation product is delivered.

The surface of the drum is divided into longitudinal sections. These sections are covered with a screen-plate which supports the filter cloth (a canvas material), the whole being wired over circumferentially to keep the cloth in place and prevent damage to it as the cake of coal is taken off by the scrapers.

Each of the sections of the drum is connected by pipes to a valve which controls the vacuum for forming the cake of coal, and also the admission of compressed air for discharging the de-watered product.

As the drum slowly revolves (one revolution in three minutes), the filtering surface is passed through the coal pulp, kept agitated to prevent settling. As each section of the drum is submerged, the vacuum acting on the pulp draws it onto the surface of the drum, where it may be seen in the form of a cake on emerging.

The liquid passes through the filter medium, whilst the solid adheres to the drum surface.

The liquid is sent to traps connected underneath to a centrifugal pump for passing the water back to the main circuit, whilst the heads of the traps are so arranged that a minimum of moisture enters the vacuum pump.

A compressor is installed and connected to the filter; and when the cake of coal is to be removed from the drum surface, the vacuum is automatically cut off and compressed air introduced which assists the discharge and cleans the filter cloth.

In the American filter, vertical leaves are employed instead of horizontal sections.

After filtering, which reduces the moisture from 50 per cent to 16 to 20 per cent, the products are taken by conveyors and added to the corresponding qualities from the other parts of the plant.

Water System:

Make-up water is fed to a main-feed reservoir connected to a centrifugal pump with a delivering capacity of 5.25 cubic metres per minute. This quantity of water is elevated to a top tank, from which it is distributed to the washing tubes, the screens, and the flotation machine.

The coal from the washing tubes is passed over preliminary drainage conveyors for de-watering prior to entering the centrifugal dryers.

This water, together with the overflow from each screen, is sent to a series of settling tanks where the slurries are tapped off and the clean water overflowed for re-use to the main-feed reservoir.

The slurries recovered are combined in a collecting pit with the coal below 16-mesh, screened out of the raw coal, and the eject water from the centrifugal dryers; this latter also containing a small percentage of solids.

Generally, all water likely to contain solids is combined with the screened product below 16-mesh, and pumped to a reservoir feeding the flotation machine.

The interposition of the reservoir is to endeavour to concentrate the solids in order to feed a pulp containing 4 parts of liquid and 1 of solids into the flotation plant, and in this the branch water pipe from the top tank also assists.

The consumption of power is as follows:

Draper plant1	motor	150	h.p.,	480	r.p.m	2.200	volts.	using	12	amn
Main feed						_,	, 02009	aoms	12	amp.
pump1	ш	80	ш	1,470	66	ш	66	"	8	cc
Conveying in-				•					U	
stallation1	"	50	"	480	и	u	46	44	no	register.
Elevating instal-									110	register.
lation1	и	50	"	480	ш	ш	"	"		66
Slurry pump1	"	50	"	480	ec .	"	ш	"	8	amp.
Flotation ma-									Ü	amp.
chine1	"	50	"	480	ш	ш	"	ш	8	и
Filter installa-									C	
tion.:1	"	150	"	480	66	"	"	ш	24	66
Flotation refuse									21	
pump1	ш	23	"	730	ш	220	ш	"	35	"
Wendell dryer1	66	50	"	480	и	2,200	ш	"	5	ш
Elmore dryer2	"	50	"	480	и	"	ш	"	8	«

The capacity of the plant is variable. For the finest washing, 30 tons per hour is obtained; for washing for washed slack, this can be increased to 40 to 50 tons per hour.

CONCLUSIONS

With the exception of flotation, all systems of washing coals rely for the most part on the separation of the coal from refuse by the difference in time materials of unequal specific gravity of equal size fall in water.

It is thus desirable for the raw coal to be efficiently screened into sizes as equal as possible prior to washing, so that the weight of the largest lightest particle is less than that of the smallest heaviest particle within the limits stated on page 18.

From a careful inspection of the washing charts, it will be seen that the results obtained from hand washing are distinctly superior to those resulting from the big installations. This is only to be expected, as the 50-lb. sample washed in the hand washing machine is subject to one hour's jigging carried out by a quick downward plunging stroke followed by a slow upward stroke. The coal is given every opportunity to settle in layers, without being hindered in such settlement by strong suction action on the return stroke. In large installa-

tions, such as No. 1 and No. 2 washing plants, the time taken for the materials to separate can be measured in seconds, scarcely one minute being required for the coal to travel the full length of the washing box. In No. 3 washing plant, with the upward current of water in the tube, if the diameter of a falling particle is A and the diameter of the tube in which it falls is A_1 , the larger the fraction A/A_1 the less the velocity with which the particle will fall.

The Baum system in No. 1 and No. 2 washing plants has been installed for some time, and, as the operators are thoroughly acquainted with their working, the results of the washing are consistent, and great regularity in production is maintained.

No. 3 washing plant was installed as an experimental plant with the idea of improving the recovery of good products, and better results have been obtained than with the Baum system, even when this latter was combined with the flotation process to treat the slurry.

The results of the washing operations with the Baum system combined with flotation, operating on Linsi special slack, show an average recovery of first-quality coal of 23.7 per cent, compared with a recovery from No. 3 washing plant of 31 per cent.

It may be added that, while the Baum system without its preliminary screening has probably reached the fullest efficiency that can be expected, it is highly probable that No. 3 washing plant will yield better results than those so far obtained.

For example, the screened products in No. 3 washing plant could be subdivided so as to give a slightly larger size for the flotation process, which would result in an increased general screening efficiency, giving better results in the washing of the 'grain' sizes. This added amount of coal, however, would be beyond the capacity of the flotation machine, as at present installed, and a study is being made to determine whether it would be desirable to instal a further machine, or

to increase the diameter of the connecting tubes communicating with the tanks and mixing chambers of the present machine so as to obtain the additional capacity.

The present yield of the 8-box machine is a maximum of 7 to 8 tons of solids per hour; and practice shows that a lower through-put is advisable for the best results.

The washing tubes give no trouble, and the same may be said of the flotation machine. In connection with this latter, it may be pointed out that it is advisable to have special linings in the stirring compartments to counteract the action of the acid.

The Wendell drying machine can comfortably deal with 40 tons of coal per hour. Owing to the coal being distributed in its different compartments, and each of these being subjected to centrifugal force, there is little loss of fine coal in the eject water, the perforated plate seeming to hold the coal, and the coal itself forming a filter bed through which the water is expelled, until the hinged gate discharges the dried product.

With the Elmore drying system, each machine can deal with 15 tons per hour. The helical scrapers, however, appear to crush the coal somewhat, and the loss of fine coal with the eject water is much larger than with the Wendell machine.

The de-watering of the flotation product in the filters caused a little trouble before modifications made the filters suitable for dealing with coal. Both of the systems in use are now equally good, although neither has yet run at the capacity it is capable of.

The American filter takes up less room, but the canvas on the filter-leaves needs renewing more frequently than that of the Oliver filter. It is possible, however, to change the leaves whilst the machine is in operation, whilst, with the Oliver filter, the changing of a cloth involves the stoppage of the plant for at least a shift and the scrapping of the wire which binds the canvas to the drum.

TABLE 6.

WASHING TEST, LINSI COLLIERY, JUNE 17th, 1926

Raw coal: Linsi special slack. Ash: 21.70 per cent. Size: 0 to 30 mm.

	- Jc										
D/C	Characteristic of stones	21.7	23.6	26.1	30.4	33.5	44.1	52.4	60.1	60.2	
D	Weight of ash cumulated upwards	2,167.3 2,099.1	2,028.8	1,867.8	1,706.0	1,616.4	1,428.5	1,281.8	983.2	492.4	
C	% in weight cumulated upwards	100.00	85.94 78.76	71.43	56.02	48.23	32.37	24.44	16.36	8.18	
B/A	Characteristic of washed coal	9.7	10.3	10.5	10.6	10.8	11.7	14.1	18.2	21.7	
В	Weight of ash cumulated downwards	68.2	218.2 299.5	379.5	550.9	642.1	885.5	1,184.1	1,674.9	2,167.3	
PX	Weight of ash in each part	68.2	79.7	80.0	9.68	91.2	146.7	298.6	490.8	492.4	2,167.3
X	% of ash of each part	9.70	11.10	10.50	11.50	11.50	18.50	37.00	00.09	60.20	
A	% in weight cumulated downwards	7.03	21.24 28.57	36.19	51.77	59.70	75.56	83.64	91.82	100.00	
Ъ	% in weight	7.03	7.18	7.62	7.79	7.93	7.93	80.8	8.18	8.18	100.00
2	Weight of parts, 1b.	2.88	3.00	3.12	3.19	3.25	3.25	3.31	3.35	3.35	40.96
1	Ref. No.	1 2	ස 4	ധ	7	∞ σ	10	11	12	13	Total:

TABLE 7.

WASHING TEST, LINSI COLLIERY, JUNE 29th, 1926

Raw Coal: Linsi special slack, No. 8 bunker. Ash: 16.00 per cent. Size: 1/16-in. to 1/8-in.

D/C	Characteristic of stones	15.6 16.2 16.0	17.6	19.4	22.1 23.9	25.8 28.5	33.1	
D	Weight of ash cumulated upwards	1,565.5	1,402.1	1,268.8	1,125.5	925.0 801.2	667.6	
C	% in weight cumulated upwards	100.00	79.83	65.41 58.20	50.86 43.52	35.78 28.04	20.18	
B/A	Characteristic of washed coal	7.0	8.2	8. 8. 8. 9.	9.1	10.6	12.1 15.6	
В	Weight of ash cumulated downwards	44.0	226.8 296.7	366.6	524.4	764.3 897.9	1,065.7	
PX	Weight of ash in each part	44.0	63.4 69.9	69.9	84.4	123.8 133.6	167.8 499.8	1,565.5
×	% of ash of each part	7.0	7.0 8.8 7.6	9.7	11.5	16.0 17.0	21.0	
A	% in weight cumulated downwards	6.29	27.38	41.80	56.48		87.81	
Ь	% in weight	6.29					7.99	100.00
2	Weight of parts, lb.	3.00	3.44 3.44	3.44	3.50		3.81	47.69
1	Ref. No.	1 2 2 2 2 2	ი 4 დ	9	& O	21	13	Total:

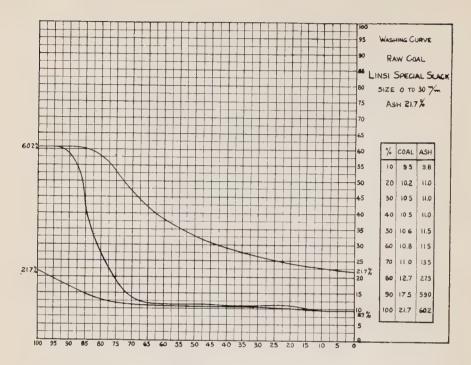


Figure 6.—See Table 6

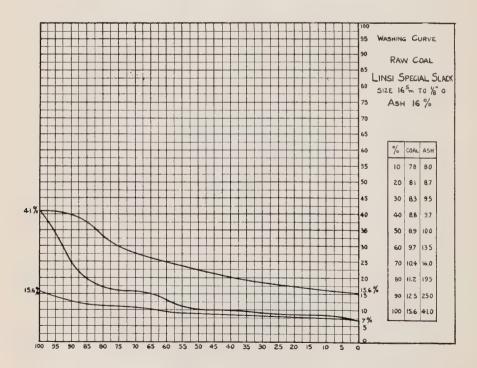


Figure 7.—See Table 7

LABLE 8.

WASHING TEST, LINSI COLLIERY, JUNE 30th, 1926

Raw Coal: Linsi special slack, No. 7 bunker. Ash: 25.00 per cent. Size: 1/8-in. to 5/32-in.

D/C	Characteristic of stones	21.3	22.3 23.3	24.9	26.6	28.6	31.2	34.2	38.2	42.5	47.1	53.5	0.09	
D	Weight of ash cumulated upwards		2,085.5		1,919.7	1,862.8	1,804.8	1,732.3	1,657.0	1,522.2	1,335.0	1,066.2	9.579	
С	% in weight cumulated upwards	100.00	93.27	79.30	72.18	65.06	57.81	50.56	43.32	35.83	28.34	19.94	11.26	
B/A	Characteristic of washed coal	7.0	7.3	7.7	7.7	7.7	8.1	8.3	9.5	11.1	13.3	16.4	21.3	
В	Weight of ash cumulated downwards	47.1	99.1	212.9	8.692	327.8	400.3	475.6	610.4	9.762	1,066.4	1,457.0	2,132.6	
PX	Weight of ash in each part	47.1	52.0	56.9	56.9	58.0	72.5	75.3	134.8	187.2	268.8	390.6	9.579	2,132.6
×	% of ash of each part	7.0	2.6	8.0	8.0	8.0	10.0	10.4	18.0	25.0	32.0	45.0	0.09	
A	% in weight cumulated downwards	6.73	13.58	27.82	34.94	42.19	49.44	26.68	64.17	71.66	90.08	88.74	100.00	
Ь	% in weight	6.73	6.85	7.12	7.12	7.25	7.25	7.24	7.49	7.49	8.40	89.8	11.26	100.00
2	Weight of parts, 1b.	3.25	3.31 4	3.44	3.44	3.50	3.50	3.50	3.62	3.62	4.06	4.19	5.44	48.31
	Ref. No.	1	N 65	74	വ	9	7	∞	6	10	11	12	13	Total:

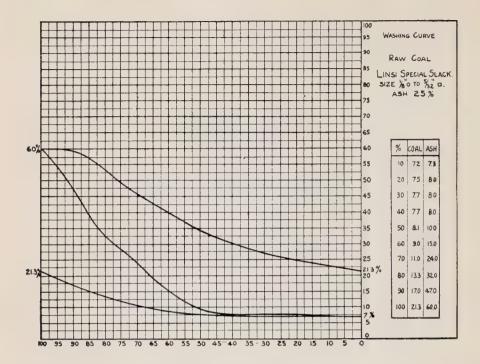


Figure 8.—See Table 8

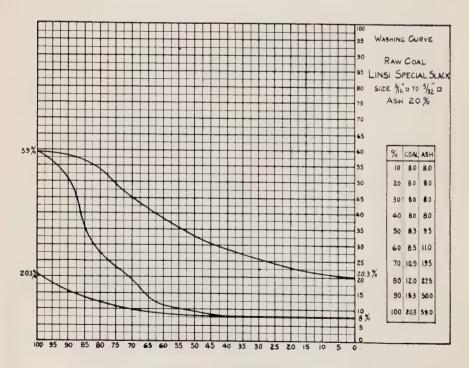


Figure 9.—See Table 9

TABLE 9.

WASHING TEST, LINSI COLLIERY, JUNE 25th, 1926

Raw coal: Linsi special slack, No. 9 bunker. Ash: 20.00 per cent. Size: 3/16-in to 5/32-in.

1)											1
D/C	Characteristic of stones	20.3	22.1	24.9	26.8	29.1	32.0	35.8	40.7	46.2	54.3	29.0	
D	Weight of ash cumulated upwards	2,027.6	1,922.4	1,809 0	1,751.1	1,692.5	1,626.2	1,547.6	1,451.6	1,296.2	1,063.8	9.629	
C	% in weight cumulated upwards	100.00	86.85	72.67	65.43	58.19	50.82	43.26	35.64	28.02	19.60	11.18	
B/A	Charac- teristic of washed coal	8.0	0.8	0.8	8.0	8.2	8.5	6.8	10.2	12.0	15.4	20.3	
В	Weight of ash cumulated downwards	50.5	161.9	276.5	335.1	401.4	480.0	576.0	731.4	963.8	1,368.0	2,027.6	
PX	Weight of ash in each part	50.5	56.7	57.9	58.6	66.3	78.6	0.96	155.4	232.4	404.2	9.629	2,027.6
×	% of ash of each part	8.0	0.8	0.8	8.1	0.6	10.4	12.6	20.4	27.6	48.0	29.0	
A	% in weight cumulated downwards	6.31	20.24	34.57	41.81	49.18	56.74	64.36	71.98	80.40	88.82	100.00	
Ь	% in weight	6.31	7.09	7.24	7.24	7.37	7.56	7.62	7.62	8.42	8.42	11.18	100.00
2	Weight of parts, lb.	3.00	3.37	3.44	3.44	3.50	3.59	3.62	3.62	4.00	4.00	5.31	47.51
1	Ref. No.	1 2	m <	# LO	9	7	∞	6	10	11	12	13	Total:

TABLE 10

WASHING TEST, LINSI COLLIERY, JUNE 21st, 1926

Raw coal: Linsi special slack. Ash: 25.00 per cent. Size: 3/16-in. to 3/8-in.

D/C	Characteristic of stones	25.4	25.6	27.9	29.5	31.5	34.0	35.1	40.5	44.8	50.1	57.3	64.7	70.0	
D	Weight of ash cumulated upwards	2,539.7	2,490.4	2,428.9	2,362.9	2,299.0	2,233.9	2,154.4	2,063.3	1,941.4	1,790.5	1,610.4	1,284.1	722.4	
C	% in weight cumulated upwards	100.00	93.42	86.84	80.00	72.90	29.69	58.44	20.96	43.34	35.72	28.10	19.84	10.32	
B/A	Characteristic of washed coal	7.5	8.4	8.8	8.8	8.9		9.5	10.5	11.7	12.9	15.7	20.2	25.4	
В	Weight of ash cumulated downwards	49.3	110.8	176.8	240.7	305.8	385.3	476.4	598.3	749.2	929.3	1,255.6	1,817.3	2,539.7	
PX	Weight of ash in each part	49.3	61.5	0.99	63.9	65.1	79.5	91.1	121.9	150.9	180.1	326.3	561.7	722.4	2,539.7
×	% of ash of each part	7.5	9.3	9.5	9.0	9.0	11.0	12.2	16.0	19.8	25.6	39.5	59.0	0.07	
A	% in weight cumulated downwards	6.58	13.16	20.00	27.10	34.33	41.56	49.04	56.66	64.28	71.90	80.16	89.68	100.00	
Ъ	% in weight	6.58	6.58	6.84	7.10	7.23	7.23	7.48	7.62	7.62	7.62	8.26	9.52	10.32	100.00
2	Weight of parts, 1b.	3.19	3.19	3.31	3,44	3.50	3.50	3.62	3.69	3.69	3.69	4.00	4.62	5.00	48.44
1	Ref. No.	-	2	က	4	5	9	7	∞	6	10	11	12	13	Total:

ABLE 11.

WASHING TEST, LINSI COLLIERY, JUNE 21st, 1926

Raw coal: Linsi special slack. Ash: 16.00 per cent. Size: 3/8-in. to 5/8-in.

	D/C	Charac- teristic of stones	8.8	7.5	18.2 19.0	0.0	1.2	9.2	9.1	0.7).1	3.7	3.6	0.0	
		g teris	1		~ # 	~	22	72	22	22	3(8	38	45	
	Q	Weight of ash cumulated upwards	1,686.8	1,634.4	1,571.9	1,443.5	1,378.7	1,312.5	1,243.0	1,166.1	1,075.4	947.9	6.687	526.0	
8 /	O	% in weight cumulated upwards	100.00	93.45	86.50 79.43	72.23	65.03	57.83	50.51	43.19	35.73	28.14	20.43	11.69	
	B/A	Characteristic of washed coal	8.0	تن د	× × • ×	8.8	&	0.6	9.1	9.5	10.3	11.3	13.1	16.8	
	B	Weight of ash cumulated downwards	52.4	114.9	178.5 243.3	308.1	374.3	443.8	520.7	611.4	738.9	6.968	1,160.8	1,686.8	
	PX	Weight of ash in each part	52.4	62.5	64.8 64.8	64.8	66.2	69.5	6.97	2.06	127.5	158.0	263.9	526.0	1,686.8
	×	% of ash of each part	8.0	0.6	0.6 0.6	0.6	9.5	9.5	10.5	12.2	16.8	20.5	30.2	45.0	
•	A	% in weight cumulated downwards	6.55	13.50	27.77	34.97	42.17	49.49	56.81	64.27	71.86	79.57	88.31	100.00	
	Ъ	% in weight	6.55	6.95	7.20	7.20	7.20	7.32	7.32	7.46	7.59	7.71	8.74	11.69	100.00
	2	Weight of parts, lb.	3.19	3.38	3.50	3.50	3.50	3.56	3.56	3.63	3.69	3.75		5.69	48.64
	1	Ref. No.		υ1 c	o 4	2	9	2	∞	6	10	I	12	13	Total:

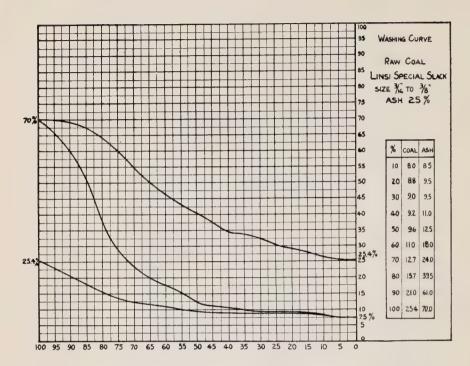


Figure 10.—See Table 10.

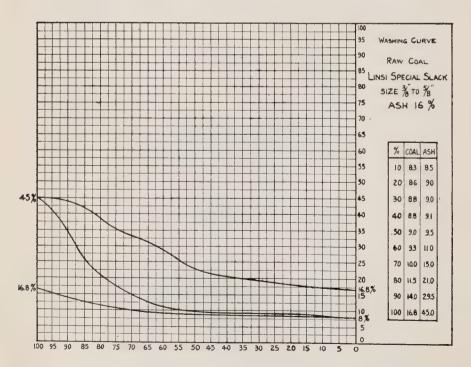


Figure 11.—See Table 11.

CABLE 12.

WASHING TEST, LINSI COLLIERY, JUNE 20th, 1926

5%-in.
over
Size:
cent.
per
70
27.
Ash:
slack.
special
Linsi
coal
Raw

2		-							
	Ъ	А	×	PX	В	B/A	C	D	D/C
Weight of	% in	% in weight	% of ash of each	Weight of ash in	Weight of ash	Charac- teristic of	% in weight	Weight of ash	Charac- teristic of
parts, lb.		cumulated	part	ب	cumulated	washed	cumulated	cumulated	stones
		downwards			downwards	coal	upwards	upwards	
3.00	6.28	6.28	8.00	50.2	50.2	8.0	100.00	2,953.1	29.5
3.00	6.28	12.56	10.00	62.8	113.0	9.0	93.72	2,902.9	31.0
3.06	6.39	18.95	10.00	63.9	176.9	9.3	87.44	2,840.1	32.5
3.19	29.9	25.62	11.00	73.4	250.3	9.7	81.05	2,776.2	34.2
3.31	6.92	32.54	12.00	83.0	333.3	10.2	74.38	2,702.8	36.3
3.31	6.92	39.46	15.00	103.8	437.1	11.1	67.46	2,619.8	38.8
3.37	7.05	46.51	15.20	107.1	544.2	11.7	60.54	2,516.0	41.5
3.37	7.05	53.56	19.50	137.5	681.7	12.7	53.49	2,408.9	45.0
3.75	7.84	61.40	23.50	184.2	6.598	14.1	46.44	2,271.4	48.9
3.21	6.71	68.11	30.50	204.6	1,070.5	15.7	38.60	2,087.2	54.0
3.94	8.24	76.35	37.50	309.0	1,379.5	18.1	31.89	1,882.6	59.0
4.06	8.49	84.84	55.00	6.994	1,846.4	21.7	23.65	1,573.6	66.5
7.25	15.16	100.00	73.00	1,106.7	2,953.1	29.5	15.16	1,106.7	73.0
47.82	100.00			2,953.1					

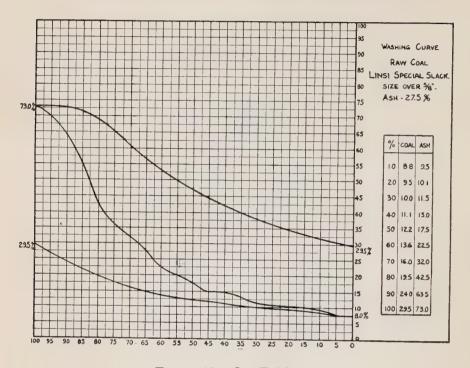


Figure 12.—See Table 12

PROGRESS IN RESEARCHES ON HEALTH AND SAFETY IN COAL MINING — 1924-27

By J. S. HALDANE AND R. V. WHEELER (Members, Inst. Min. Eng.)

(Montreal, Que., Meeting, August 23rd, 1927)

During the three years since the First Empire Mining and Metallurgical Congress, at which a series of papers dealing with health and safety in coal mining was presented, substantial progress has been made in Great Britain in research on these problems. To a large extent this progress is due to the activities of the Safety in Mines Research Board and its Committees. In this connection, the outstanding event as regards the organization of research has been the completion of a scheme of cooperation between the Safety in Mines Research Board and the United States Bureau of Mines. This scheme is a comprehensive one, involving as it does the unrestricted exchange of information between individual investigators regarding researches in progress, the preparation of joint programmes of research, and the interchange of personnel. In this way the solution of some of the many problems which both organizations are attempting to solve will be hastened; whilst opportunity is afforded for testing the validity of conclusions that may have a far-reaching effect on the mining industry under a greater variety of conditions than are available to either organization singly. The scheme has already proved highly beneficial to both groups of investigators, in increasing their efficiency and widening their interests, and its value will undoubtedly increase as time goes on.

The Safety in Mines Research Board is the central organization for research on problems connected with safety in coal-mining in Great Britain. The work of the Board is financed almost wholly from the Miners' Welfare Fund (a fund raised by a statutory levy on the coal mining industry), and, as an organization within the industry itself, the Board

should command the whole-hearted support, whilst receiving the friendly criticism, of all engaged in the mining industry.

The Board's researches are carried out mainly at its two Research Stations, at Harpur-Hill near Buxton in Derbyshire, where large-scale experiments, for example coal-dust and fire-damp explosions (which have been transferred from Eskmeals in Cumberland), are conducted; and at Sheffield, where most of the laboratory work is concentrated. Engineering researches are, for the most part, carried out at the Imperial College of Science and Technology, South Kensington; whilst the Board is in close touch with (and partly finances) the work of the Mining Research Laboratory of Birmingham University, the Lancashire and Cheshire Coal Research Association, and a number of individual investigators at different universities. On certain special health questions with reference to mining, the Medical Research Council has initiated research.

A review of progress in researches in Great Britain on health and safety in coal mining can conveniently be drawn from the Annual Reports of the Safety in Mines Research Board, which contain each year summarized accounts of all researches in which the Board is concerned. Details of these researches are published mainly in a special series of papers issued by the Board and partly in the *Transactions* of the Institution of Mining Engineers, the *Journal* of the Chemical Society, the *Journal of Hygiene*, and similar publications.

HEALTH

Problems relating to the health of the miner can be grouped according as they are concerned with gases encountered in mining practice; with heat in mines; with dust in mines; with defective illumination; or with underground infection.

Noxious Gases

From a physiological point of view, most of the gaseous impurities met with in mines under normal conditions are so trivial in amount (except, perhaps, after blasting) as to be

hardly worthy of consideration; whilst there is little to add regarding the study of the few important impurities, sometimes present in large quantities, such as fire-damp and blackdamp.

Heat in Mines

It is essential to the well-being of the miner that the temperature of his body should remain constant within narrow limits, and an increase of more than a degree or two above the normal may cause serious physiological disturbance. In consequence, the increasing depth at which mines are being worked, tending to entail a corresponding increase in the temperature of the atmosphere, has introduced a difficult

problem.

The study of the control of atmospheric conditions in deep and hot mines has been continued in connection with a Committee of the Institution of Mining Engineers, at the Mining Research Laboratory of Birmingham University and at a number of collieries. Investigations are being carried out concerning the possibilities of cooling and drying the air by local refrigeration plant. Progress has been made in the further development of Professor Hill's Katathermometer, the wet-bulb thermometer, and delicate anemometers, as indicators of atmospheric conditions in hot places underground where the air is almost stagnant. A great deal of spade-work is required and is being carried out with a view to elucidating the thermal changes in, and the various causes controlling, underground air temperature and humidity and the physiological problems connected with work in atmospheres of high temperature and humidity. The influence of refrigerating plants installed underground, of ice distributed locally, and of purely local arrangements for increasing the velocity of air-currents are, for instance, being studied. The further these investigations have been carried, however, the more clearly does it seem to emerge that, as a general rule, it is extremely difficult to control temperature and moisture by any other means than abundant ventilation directed on the parts of a mine which are being worked. The mine requires to be laid out, and the air-currents controlled, with this end in view; and provided that this condition is fulfilled, there appears to be no difficulty in controlling the conditions as regards temperature and moisture up to the greatest depths at present reached in mining enterprises. In a climate, however, where the atmosphere is liable to become both warm and very moist, it may be necessary to dry the incoming air by refrigeration or other means.

Investigations on acclimatization to warm and moist atmospheres, and to the serious, but easily avoidable, physiological effects of loss of chlorides by the body through sweating, are being continued.

Dust in Mines

The work of coal mining nearly always entails the inhalation of considerable quantity of dust. So long as that dust consists only of coal, soft shale, or certain other kinds of stone or inorganic material, and the amount inhaled is not excessive, its inhalation seems to have no harmful effect. There was a fear that, when the practice of stone-dusting for the prevention of coal-dust explosions became general, the use of an excess of shale dust (which may contain about 35 per cent of free silica) might prove harmful. This fear seems to have been groundless. Suitable shales for grinding into dust are not, however, always readily available locally and there may be a danger of dusts being introduced into the mine which may render men who constantly breathe them liable to phthisis. This matter is receiving watchful attention.

Clear evidence has recently been obtained that miners' phthisis exists as an occupational disease among certain classes of coal miners, although of rare and occasional occurrence. At present there is no evidence to show that any miners are liable to contract it other than those constantly engaged in driving roads with machine drills through highly silicious rock, such as sandstone, without effective means of dealing with the dust created; but as the danger incurred in such work is a very real one, much attention has been given to the means of meeting it. A dust-trapping device which appears to be perfectly effective has been devised and tested; and it is hoped that this will meet the danger. Experiments on the effects

in animals of inhalation of various kinds of dusts are also being continued and have furnished very instructive results. No signs of injury to the lungs by dust-inhalation have been discovered in the bodies of a large number of pit ponies after work for many years on stone-dusted roads.

Underground Infection

Ankylostomiasis, or miners' anæmia, is, perhaps, the only specific disease by which the coal-miner is liable to be infected. Though this disease was found about 25 years ago to be very serious in Cornish tin-mines it was stamped out there by simple hygienic measures, and coal mines in Great Britain have remained free of it. A few cases of spirochaetal jaundice from underground infection in one or two shallow Scottish collieries have recently been discovered. The infection seems to have been carried by water from the surface. The form of septic cellulitis or bursitis known to coal-miners as 'beat-knee', 'beat hand', and 'beat elbow', have been investigated by the Medical Research Council, and the precautions required for prevention explained.

Miners' nystagmus is a disease of the eyes which entails much suffering and puts a heavy economic burden on the coal-mining industry. Several theories have been put forward as to the cause of the disease, such as the attitude assumed by the coal hewer when at work, carbon monoxide poisoning, or the disturbance of the nervous system by some toxin. There is now no doubt, however, that the true cause of miner's nystagmus is defective illumination; and from this standpoint the improvement of lamps, referred to later, is a matter of great importance.

SAFETY

The main subjects of research on safety in mines can be classed under the following heads: Coal dust; fire-damp; spontaneous combustion; electricity; explosives; safety lamps; support of workings; wire ropes and other mechanical appliances.

Coal Dust

A most important, if not the most important, subject of research in relation to safety in coal mining is the prevention of coal-dust explosions. The use of stone-dust with this object has, up to the present, afforded the most satisfactory results, the chief problem being to determine the requisite amount of stone dust to be mixed with different coal dusts in order that the mixtures shall be incapable of propagating flame.

The results of independent investigation in England and in America were not entirely in agreement, and one of the earliest arrangements under the scheme of co-operation between the two countries was the carrying out both at the Experimental Mine at Bruceton, Pennsylvania, and at the Experimental Station at Eskmeals, Cumberland, of a series of tests with a standard British coal and standard American coal under the conditions prevailing at each station. It was concluded that there was sufficient uniformity in the results to justify the direct application to British coals of the results of the numerous series of experiments made at the Experimental Mine in America as regards the effects of such factors as the degree of fineness of the coal dust and its chemical composition. An account of these co-ordinating tests formed the first joint publication by the Safety in Mines Research Board and the United States Bureau of Mines.

Another important conclusion having direct application to actual coal mining conditions was again the result of joint work, inasmuch as one of the Bureau of Mines' investigators was detailed to work at the Eskmeals Experimental Station during the summer of 1924. It was found, from experiments in the 7½-feet diameter gallery there, that there is less danger of a coal-dust explosion developing from a given source of ignition at a long-wall face than at a 'dead-end' or *cul-de-sac*; and, generally, that branch roads near the point of ignition of an incipient coal-dust explosion, by affording release of pressure, retard and may prevent the development of the explosion. Again, the course of a partially developed explosion may depend upon the arrangement of the mine roadways and branching passages, a knowledge of which will indicate

the positions in the mine at which the most stringent measures should be taken against the accumulation of coal-dust or the occurrence of a source of ignition.

The most important physical quality of a coal dust with respect to its ignition and, when in the form of a cloud, as a medium for the propagation of flame, is its degree of fineness. Methods of determining the degree of fineness of very small particles have therefore formed the subject of close study. It would seem that air elutriation methods provide the most satisfactory means of obtained grades of dust of definite range of specific surface for the purpose of correlating specific surface and degree of inflammability of a dust cloud.

Although the degree of fineness of a coal dust mainly determines the ease with which it can be ignited and can propagate flame when raised as a cloud in air, it is not the only factor on which its inflammability depends, but its chemical properties, which render it of greater or less reactivity towards oxygen, must be taken into consideration. Experiments have been made to determine the effect, apart from any other factor, that the chemical composition of a coal dust has on its inflamma-The results disclose a relationship between the inflammability of a dust and its content of 'volatile matter' as determined by standard methods, the more inflammable dusts containing the higher proportions of volatile matter. It is not suggested that the contents of volatile matter of a coal directly determines the inflammability of its dust. In view of the fact that much of the volatile matter of coal is incombustible, such a contention would be hard to maintain. amount of volatile matter may, however, give an indication of the relative proportions of those ingredients of the coal which play the most active part in the phenomena of the rapid combustion of its dust.

Fire-damp

The objectives of research on the ignition of fire-damp with reference to the requirements of the coal-mining industry are: To ascertain the conditions under which fire-damp will ignite, in order that means may be taken to avoid their occurrence in the pit; and to ascertain which are the most readily ignited mixtures of fire-damp and air, in order that these mixtures may be used when determining the liability of colliery equipment, such as flame and electric lamps, electrical apparatus generally, and explosives, to ignite fire-damp.

When an inflammable mixture of fire-damp and air is raised to, and maintained at, a temperature higher than the 'ignition temperature', it becomes ignited — but not immediately. There is a certain interval, termed the 'lag on ignition', the length of which depends upon the temperature and upon the composition of the mixture. This 'lag on ignition' has an important bearing on safety in coal mines, as it accounts for the fact that it is very difficult, if not impossible, to cause an external ignition of fire-damp by the gauze of a miners' flame safety lamp heated by fire-damp burning within it; and it provides ground for the hope that it may be possible to compound explosives, the flame from which, being of exceedingly short duration, might not be able to ignite fire-damp despite its high temperature. The intervention of flame, however, is not essential for the ignition of an inflammable gaseous mixture. A fire-damp-air mixture can be ignited by pressure alone, and this fact may have an important bearing on the question of the ignition of pockets of gas in crevices by the pressure produced when a shot is fired, even were it possible to suppress the flame of the explosive entirely.

A matter which does not appear to have been investigated hitherto is the extent to which flame can be projected beyond the region in which an inflammable mixture originally existed. Experiments have shown that the projection of the flame of a fire-damp explosion into air is between five and six times the length of the original column of explosive mixture.

A mixture of fire-damp and air is only inflammable when the fire-damp present lies between certain percentages usually referred to as the lower and upper 'limits'. These limits are modified by the direction that the flame can take (upward, downward, or horizontal), by the presence of uninflammable gases and of water vapour, by temperature and pressure. The effects of each of these variables have been studied. Particular attention has been directed to the estimation of the effect of the presence of black-damp in a mixture of firedamp with a mine atmosphere. This investigation has also covered the question of the extent to which the introduction of carbon dioxide alone can render a fire-damp explosion impossible. The results of the investigation can be summarized as follows: No self-propagating inflammation is possible in any mixture of methane with atmospheres consisting of air mixed with (a) 25 per cent or more of carbon dioxide, or (b) 38.5 per cent or more of nitrogen. The amount of black-damp that must be mixed with air to form an extinctive atmosphere for methane flames lies between these percentages, according to the proportions of nitrogen and carbon dioxide in the particular sample of black-damp.

Spontaneous Combustion

The liability of certain coals to spontaneous heating may constitute a considerable source of danger in coal-mining, inasmuch as it is a potential cause of underground fires and, thereby, of fire-damp and coal dust explosions. The spontaneous heating of coal was at one time believed to be due solely to the presence of iron pyrites, but it is now established that, although the presence of pyrite may sometimes be a contributory or even a determining factor, oxidation of the coal substance itself is the most frequent cause. This being so, it is important to find out what are the most readily oxidizable constituents of coal. Ouite recent researches carried out by the Safety in Mines Research Board have shown that the portion of the coal conglomerate that must be held responsible for spontaneous combustion is the 'ulmin' portion, which constitutes the bulk of the substance of bituminous coals, particularly of bright coals.

Research is in progress regarding the liability of coal that has been heated to a fairly high temperature (200-300°C.), and then allowed to cool, to oxidize rapidly on exposure to air, a matter of importance in connection with the reopening of sealed-off areas in a mine. It would appear that such treatment renders some coals more liable to self-heat than before.

The study of the control of gob-fires will, it is believed, be advanced by the recent installation, at the Safety in Mines Research Station at Buxton, of a chamber simulating a mine goaf surrounded by ventilation roads. Here a study is being

made of the progress of heating in coal behind stoppings, more particularly as regards changes in the atmosphere which may result in a gaseous explosion. The early detection and control of heatings in the mine by means of systematic analyses of the return air has been further developed, particularly by determination of the ratio of carbon monoxide formed to oxygen consumed. The effects on gas evolution in a mine of reversal of the ventilating current is also being studied, since, in the event of a fire reversal may be desirable. Study has also been made of the effect of change of barometric pressure on the feeding of sealed-off fires with oxygen.

Electricity

The safe use of electricity underground is dependent largely on the provision of flame-proof enclosures for the electrical machinery. Three devices for affording release of the pressure of a fire-damp explosion within a casing, without permitting the passage of flame, have been developed, the so-called 'flange-protection', 'perforated-plate protection', and 'ring-relief protection.' Flange protection provides for the release of the pressure at flanged joints, the spacing between the flanges, and their width, being so proportioned that flame cannot reach the outer atmosphere. The purpose of the perforated-plate and ring-relief devices is to enable additional release of pressure to be obtained from large casings, where the area at flanged covers may be small in proportion to the volume of the enclosure. The perforated-plate device, as its name suggests, embodies perforated sheet metal the specification for which was determined by experiment: The ring-relief device consists of a number of metallic rings, separated one from another by distance pieces, assembled in the form of a cage and held securely in position by a circular end-plate and bolts connecting the end-plate with the casing. The interior of the casing thus communicates with the outer air through the series of gaps formed by the rings and the faces of the distance pieces.

Explosives

The safe use of explosives in coal mines is, by the very nature of the substances used, one of the most difficult problems in research on safety in mines. It is clearly necessary to safeguard both the selection and the manner of use of explosive compounds where inflammable gas or dust may be present. In order to obtain a measure of the ability of an explosive to cause the ignition of gas or coal-dust, a gallery test has been devised in which mixtures of fire-damp and air are used as the inflammable medium into which the explosives are fired. Unexpected difficulties were encountered in the working out of this test, the results being apparently affected by factors of an indeterminate character. Whilst a test of this character will serve as a rough guide in researches on the safe use of explosives, it is clear that most reliance must be placed on work of a fundamental nature, progress in which is, unfortunately, very slow.

Typical of the fundamental studies now being carried out regarding the behaviour of explosives, are those with respect to the shock-wave sent out when an explosive is detonated. Experiments in which simultaneous records were obtained of (a) the rate of propagation of the shock-wave and pressure-waves, (b) the rate of projection of the flame and products of detonation, and (c) the relative positions of each after the shot has been fired, have shown that the flame of a detonator never comes into contact with the external atmosphere, being always surrounded by a 'blanket' of incombustible gases (the products of detonation), a fact which explains the inability of the flame of a detonator to ignite mixtures of fire-damp and air.

Safety Lamps

During recent years the lighting efficiency of flame safety-lamps has been considerably improved as the result of systematic study of the factors involved. A large number of small details which might be considered insignificant are really of considerable importance as regards their effect on the light that a flame lamp can give. The proportions (particularly the length) of the gauzes, the area of the ventilation holes, and the type of burner, all require attention. With suitably proportioned lamps it has been found possible to obtain a candle-power of 1.5 with the ordinary Marsaut, of 2.0 with bottom-feed lamps, and of nearly 3.0 with lamps of the 'combustion-tube' type.

Progress as regards the provision of electric hand-lamps of high candle-power and reasonable weight has not been so satisfactory; nor can it be said that success has attended efforts to produce a reliable fire-damp detector for incorporation in a miners' electric hand-lamp—a desirable, if not a necessary adjunct.

Support of Workings

About 50 per cent of the deaths due to accidents in mines in Great Britain are occasioned by falls of ground. The loss of life and the injuries due to this cause have continued with little variation for many years. The subject is one which, in some of its most important aspects, does not lend itself to experimental study, the only safeguard against many of the accidents due to fall of roof and sides being experience and judgment on the part of the management, and on the part of the men carefulness and response to discipline. There are, however, some features of the problem which admit of measurement and efforts are being made to deal with these. Arrangements have been made for measuring, by means of dynamometers, the earth pressures experienced underground, and the strengths of different types of props and roof supports are being measured.

A systematic survey is being made of the different methods of timbering and of roof support used throughout the country and abroad, as a result of which it is hoped that knowledge of the most successful methods may be disseminated and the proportion of accidents due to falls of roof thereby reduced.

Mechanical Appliances

Accidents, too, frequently occur through the breaking of winding and haulage ropes. It is satisfactory, therefore, to be able to record that researches are in progress having for one of their main objects the foretelling when risk of failure of a rope may be anticipated. These include investigations of the corrosion-resisting properties of special steels, of the changes produced in the physical properties of steel wire subjected to galvanizing and other processes intended to resist corrosion, and of methods for rapidly detecting deterioration in wire ropes. Similarly, as regards arresting runaway tubs,

close study is being made of the most promising methods. It is too early yet to mark progress in either of these branches of study.

In preparing this brief survey of progress in research on health and safety in coal-mining, we have been assisted by Mr. Allan Greenwell, Librarian of the Safety in Mines Research Station, whose help we wish gratefully to acknowledge.

ASBESTOS MINING AND MILLING

By J. G. Ross (Member, C. Inst. M. & M.)*

(Quebec, Que., Meeting, September 5th, 1927)

INTRODUCTION

Asbestos, a mineral known and used in ancient times, has become an article of commerce only during the life of the older asbestos miners in Quebec. The diversified uses of this adaptable substance are now numbered by the hundreds. Wherever insulation or resistance to heat may be required, asbestos, either by itself or mixed with some other material, may be employed to advantage. Regarded as a curiosity for centuries, it is now in such common use that it is familiar to all.

The demand for long fibre for use in the manufacture of textiles and brake-lining, and for shorter fibre for making shingles, paper, and insulation materials, serves to support the industry of mining asbestos-bearing rock and milling the fibre from it in Canada, South Africa, Russia, Cyprus, and the United States.

The commercial use of asbestos dates only from 1876, in which year large deposits were discovered in the Province of Quebec, Canada. Prior to that year a small amount of long fibre amphibole asbestos was produced in Italy.

OCCURRENCE IN CANADA

In the Eastern Townships of Quebec, along the line of the Quebec Central railway, is an area of serpentine rock. It extends westerly into Vermont and easterly to Gaspé. A part of it, the section from Danville to East Broughton, contains areas in which occur deposits of commercial asbestos fibre of the chrysotile variety. Pits have been opened on some of these deposits at Danville, Coleraine, Black Lake, Thetford Mines, Robertson, and East Broughton. Thetford is distant 68 miles from Quebec city, East Broughton lies 25 miles to the northeast of Thetford, while Danville is 45 miles to the southwest of Thetford.

^{*}Consulting Mining Engineer, Milton Hersey Company, Limited, 84 St. Antoine Street, Montreal.

From these districts a greater part of the asbestos for the manufacturing plants of America and Europe comes. A few years ago Canada produced about 80 per cent of the fibre consumed. At the present time, however, while practically all the lower grades come from Canada, about 60 per cent in value of the longer grades is brought from Rhodesia, with a small amount from other countries.

GEOLOGY OF THE PRODUCING AREA

The following description of the main geological features of the asbestos-producing district of the Eastern Townships of Quebec is quoted from a recent publication of the Geological Survey of Canada (1):

"The asbestos deposits occur in altered peridotite which, with other closely associated, related intrusive rocks, forms a comparatively narrow, interrupted band extending from the Vermont border just west of lake Memphremagog for 150 miles to the northeast. The zone of igneous rocks, frequently referred to as the 'serpentine belt', lies along the eastern side of the westernmost of the three ranges which in Quebec compose the Appalachian region. The western range has an anticlinal structure, so that in a general way the strata into which the igneous rocks have been intruded, dip to the southeast. In most places the invading igneous bodies occur along the eastern margin of a belt of slates, quartzites, and sandstones of Cambrian or, in part, early Ordovician age.

"These rocks, with occasional areas of the next overlying series, are mostly foliated, in places are schistose, and in many places are intricately folded and wrinkled. The earlier Palæozoic strata are bounded on the southwest by younger dark slates with a conglomerate at their base holding fragments of the older strata to the west. The dark slates are considered to be of Trenton age and, in places, are cut by the igneous bodies which are older than the main period of orogenic disturbance of late Devonian time and may be Ordovician.

"The igneous rocks are in general confined to a zone less than a mile wide. The individual bodies have the forms of inclined sheets, in some cases outcropping with widths of 500

⁽¹⁾ Geology and Economic Minerals of Canada, by G. A. Young; Geological Survey of Canada, Economic Series No. 1, 1926, pp. 100-104.

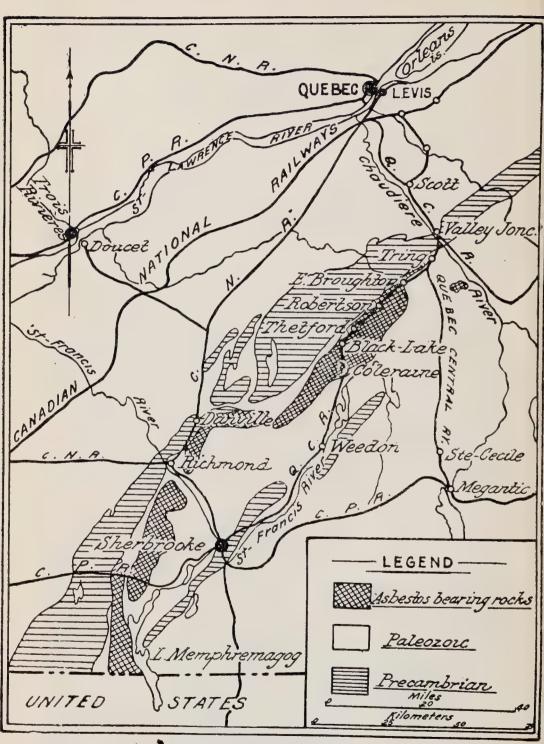


Figure 1.—Sketch map of the "Serpentine Belt", Quebec, in which occur the deposits of Asbestos.

to 1,000 feet and with lengths of several miles or more; or they are stock-like masses, in some cases occupying areas of 10 or more square miles; or they appear to be large, thick, lenticular, laccolithic-like bodies. In some instances, two or more sills parallel one another and in the important Black Lake-Thetford district, the igneous masses for a few miles form two nearly parallel belts 2 miles apart and each 2 or 3 miles broad. The igneous rocks form a series consisting of peridotite, pyroxenite, gabbro, and porphyrite with which are associated small bodies of granite and aplite, and vein-like, granular aggregates of various minerals for the most part alumino-silicates, such as diopside, vesuvianite, and garnet.

"In the sills where two or more of the igneous rocks are present, they occur, from the base upwards, in the order—peridotite, pyroxenite, gabbro, and porphyrite, and where two sills parallel one another, the upper consists in part or wholly of members higher in the general order than those forming the lower sill. In many of the stock-like and laccolithic-like bodies, the central or deeper portion appears originally to have been composed of peridotite and the outer or upper portion of higher, more acid members of the series arranged in regular order. Depending on the present attitude of the mass and the depth to which erosion has proceeded, the body at the surface may now consist mainly of one rock type or of one rock type partly or completely surrounded by one or more zones which outwardly grade from basic to more acid.

"Where two or more members of the peridotite-porphyrite series are present, they generally merge into one another, and in such instances it is apparent that differentiation has proceeded in situ. But all members of the series are not always present in the same body. It is particularly noticeable that, both towards the southwest and northeast ends of the serpentine belt, the igneous bodies are largely or solely fine-grained gabbro or diabase. Furthermore, in various places more acid members cut more basic; or finer-grained facies of the same rock type cut coarser facies. The granite, which generally forms comparatively small, irregular, or dyke-like bodies, and the aplite dykes, distinctly cut the other members of the assemblage. It is apparent that partial or complete differentiation in some cases took place prior to the injection of the igneous masses.

"The asbestos of the Thetford-Black Lake area occurs in narrow, vein-like bodies traversing partly altered peridotite which outcrops over the greater part of a deeply eroded mass of laccolithic form about 3 miles wide and a few miles longer. The asbestos 'veins' are specially numerous in local areas. Most of them are less than one-half inch in width, but 'veins' 2 to 3 inches wide are not uncommon and some have been found 5 or 6 inches wide. The asbestos is of the flexible. chrysotile variety and the fibres are disposed at right angles to the walls of the 'veins'. As a rule the individual fibres do not extend across the complete width of a 'vein', but reach from both walls inwardly to a parting frequently defined by a film of iron oxide, usually magnetite. The asbestos 'veins' are sharply defined and are bounded on both sides by a zone of peridotite completely altered to serpentine of essentially the same composition as the asbestos. The bounding serpentine zones are sharply defined from the adjoining wall-rock, though it too is partly altered to serpentine, and the degree of serpentinization gradually decreases outwardly. The combined width of the asbestos 'vein' and the two bordering, completely serpentinized zones bears a constant relation to the width of the asbestos 'vein' and is in the proportion of 6.6 to 1.

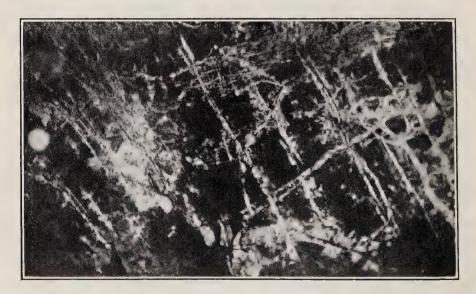


Figure 2.—Black Lake—Thetford district, Quebec. Showing the extraordinary number of 'veins' of high-grade asbestos.

(Flashlight photograph taken in a drift).

"The asbestos 'veins' with their borders of pure serpentine occur at all attitudes from horizontal to vertical, but in any one place the more prominent 'veins' form a rudely rectilinear system corresponding to the main joint system present in the peridotite. Such 'veins', with widths up to 2 inches or greater, may extend for 100 feet or more. Other 'veins', generally narrower and more irregular, intersect these and themselves at various angles. In places, the asbestos 'veins' are very numerous; seventy have been noted in a width of 2 feet.

"The small bodies of granite and aplite which accompany the other members of the serpentine belt, and which are presumably an extreme differentiation phase of the one magma, are more numerous in the vicinity of the productive asbestos areas than elsewhere. This relation may mean that the narrow serpentine bands, with their median zones of asbestos, were produced by solutions or vapours derived from or accompanying the granitic bodies, and that these solutions or vapours traversed fractures now indicated by the partings of iron oxide contained in the asbestos 'veins'. Furthermore, the partings may have been formed at the time of the granitic intrusion.

"In the East Broughton district, 25 miles northeast of the Thetford-Black Lake district, the asbestos occurs in a different In this northeastern district, the intrusives form comparatively thin sills, mostly 100 to 600 feet thick. lower portion consists of serpentine carrying little asbestos. Above this zone, in places, is greatly fractured serpentine in which asbestos is abundantly developed in thin layers of overlapping fibres which lie parallel to, and along, the numerous planes of fracture. In places, almost the whole rock is fibrous. The asbestos-bearing zone in some cases is surmounted by a band of greenish schist representing a highly metamorphosed part of the sill, or such rocks may form a separate sill lying stratigraphically a few hundred feet higher than the asbestosbearing sheet. Talc, in veins, and soapstone, in large masses, occur in the asbestos-bearing part of the sill or above it. The serpentine of these sills appears to have been derived mainly or solely from pyroxenite."

CHARACTER OF THE ASBESTOS

Cross-fibre, the more valuable variety, is mined at Thetford and Black Lake; banded cross-fibre at Coleraine; slip-fibre at East Broughton; and both cross and slip at Robertson. While the fibre, as found, appears to be solid rock, it may be pulled apart in fine shreds which resemble silk threads. In the mass, loose fibre resembles carded wool. The local name applied by the French-Canadian miners is 'cotton'.

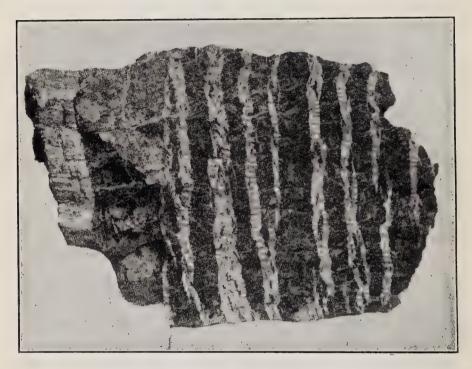


Figure 3.—'Veins' of cross-fibre asbestos in massive serpentine.
Black Lake—Thetford district, Quebec.

Although, like silk or cotton, asbestos may be spun, it spins with more difficulty than these, owing to the fact that the individual threads are smooth like glass, whereas vegetable or animal fibres are provided with protrusions or hooks which enable them to be bonded together.

Chemically, the fibre is similar in constitution to the enclosing serpentine rock, being a hydrous magnesium silicate, in which the percentage of water is somewhat variable. Usually the softness or workability of the fibre is directly proportional to the water content. Thetford fibre contains 12 per cent

water and is the most valuable, while that from Coleraine, containing only 9 to 10 per cent water, is harsh and unsuitable for the best textile purposes.

Apart from the fact that it can be spun, asbestos depends for its commercial use on its high resistance to heat and chemicals, and its insulating quality as a non-conductor of heat, cold, and electricity. It can withstand, when in the mass, temperature up to 800° or 1,000° F. without disintegrating. For some purposes it is used alone and for others it is mixed with cheaper materials.

Uses of Asbestos

Asbestos spun into yarn, sometimes mixed with cotton for greater ease in working, or strengthened by fine copper wire, is used as a basis for all asbestos textiles. The yarn is woven

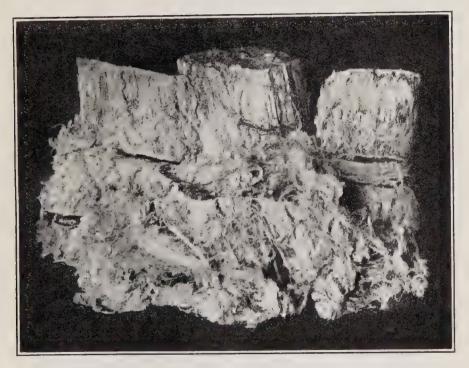


Figure 4.—Chrysotile-asbestos from Thetford Mines, Quebec. Quality "Crude No. 1".

into fabrics of various kinds, weights, and thicknesses, which are then made into gloves, clothing, theatre curtains, and many other useful articles. The cloth coated with rubber, or impregnated with graphite, is fashioned into gaskets and steam packing. Very fine fibre forms filaments in electric lamps and filters in chemical laboratories. For the motor industry, seventy million feet of brake-lining is made of asbestos annually in the United States alone, fifteen million feet for new cars and the balance for replacement. A lower grade of fibre, mixed with cement, makes asbestos shingles and lumber. Mixed with magnesia and other cementing materials, it is manufactured into pipe and boiler coverings, in the shape of blocks, tubes, or felt.

While the bibliography on asbestos is extensive, published information on present practice in mining and milling from a technical standpoint may be said to be non-existent. A Department of Mines *Memoir* (¹), by Fritz Cirkel, first published in 1905 and revised in 1910, but now out of print, is still regarded as a standard hand-book on the industry.

Visitors to the asbestos-producing districts of Canada and Rhodesia are impressed by the extent of the workings and the importance of the industry, yet can secure but little information on the technical features peculiar to it.

With the present tendency toward centralized control, there is an even more marked disposition on the part of the operating companies to withold information which may be of use to possible competitors. An air of secrecy has always surrounded asbestos milling operations, and it is not even considered good form for the superintendent of one plant to study the mill of a fellow superintendent working for the same company.

Asbestos is a mineral of mystery to the laity, and, even with the widespread use of the great variety of articles manufactured from it, the general public does not altogether realize that it can be a rock product. The air of mystery pervades the methods of recovering and preparing the fibre, and the consequent lack of co-operation has hindered proper development of processes. This is in striking contrast to conditions in metal mining and refining.

⁽¹⁾Chrysotile-Asbestos: its occurrence, exploitation, milling and uses, by Fritz Cirkel; Mines Branch, Department of Mines, Ottawa, Report No. 11 (1905) and No. 69 (1910).

Figure 6.—Thetford Mines, Quebec. Loading asbestos rock by hand into boxes, which are lifted to cars by steam crane.



The Canadian field, being the pioneer, has been the scene of the gradual evolution of methods of lifting the rock, separating the fibre, and preparing it for the use of the manufacturer. Other countries which have progressed beyond simple hand cobbing of crude fibre have adopted some form of Canadian practice so far as they could learn it.

MINING METHODS

In the mining of asbestos-bearing rock, standard methods, applicable to taking out rock containing any mineral, are in use. These vary only with the topography, location, and extent of the deposit. The most economical system for any given deposit is, however, not always employed. Once a



Figure 7.—Thetford Mines, Quebec.
Pit in foreground worked by cable-way hoists.
Pit to left: cranes and steam-shovel working pit face.
Pit to right: large cable-way derricks.

method has proved successful in one location, it is often copied in another where a more economical system would suggest itself to an engineer or experienced operator from an allied field. Choice of mining methods is, however, limited by the character of the mother rock, peridotite or serpentine. These are generally fractured in all directions by 'dry heads', or slickensided planes, so that, once a face is opened, the rock

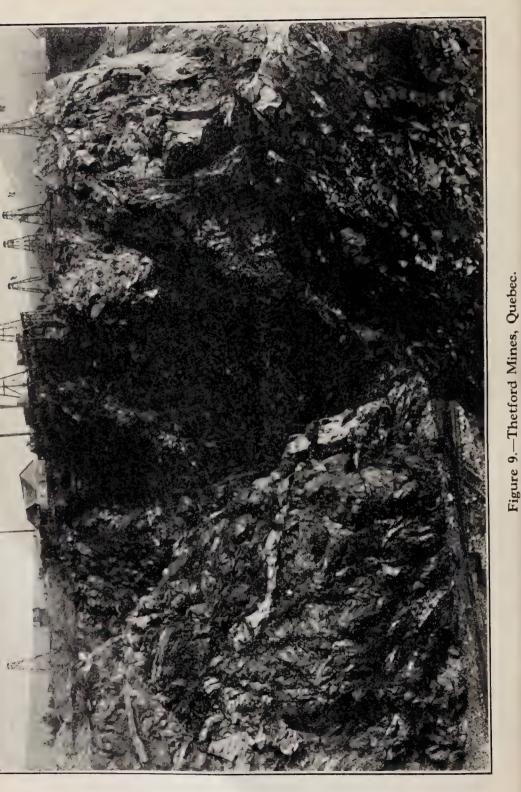


Figure 8.—Thetford Mines, Quebec. Foreground: railway tracks on pit floor. Background: cable-way towers and mill.

has little cohesion and it is inadvisable to open large stopes, or to leave steep faces exposed on an open pit. The low value per ton of rock prohibits the use of methods requiring expensive timbering.

In Canada, quarrying is the popular method. Development has been largely in the hands of the original prospectors, who first worked open pits during the summer months for the recovery, by hand cobbing, of crude fibre only. Hoisting from the pits was by means of small boom derricks.

As the depth of the pit increased, larger boom derricks were installed and more of them were placed around the circumference of the pit. Then, as the opening widened out, the boom derricks were replaced by cable-way hoists. With the continued growth in size of the pits, the cable-way hoists



have been elaborated and hoisting mechanism refined. Today, the largest hoist has a span of 1,400 feet, with a carriage running on a 3-inch cable, and it lifts at one time a load of 10 tons of rock from a depth of 450 feet. The steel derrick supporting the cable has a height of 105 feet, while the electric motor for hoisting requires 450 h.p.

In the East Broughton section, the rock breaks readily from the face. The serpentine itself is fibrous to such an

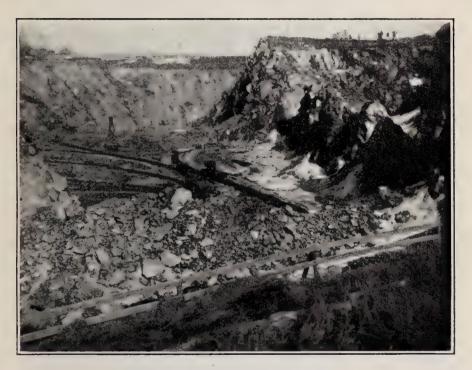


Figure 10.—Thetford Mines, Quebec.

Foreground: floor of pit, with tracks for transportation of asbestos rock, loaded by cranes and steam-shovel.

At right: party wall to pit, served by cable-way hoists.

Background: cable-ways over deep pit.

extent that up to 20 per cent of the rock mined may be milled into merchantable fibre. The asbestos is the variety known as 'slip', and practically no crude or cross-vein fibre is found. But little drilling and blasting is required to loosen the rock on the face, and it is easily loaded into cars by means of a steam-shovel. Rock may be so won at a comparatively low cost—as it necessarily must be, the grades of fibre produced being low in value.

In the central, or Thetford, area the open pits are worked in benches. Horizontal holes are drilled into the bottom of the bench by means of standard hammer drills of various makes to depths of from 12 to 16 feet. Sufficient 40 per cent dynamite is loaded into the holes to shatter the rock into sizes which can be handled by labourers. After a face has been drilled and fired, a supply of rock results which is ample to keep the loaders on one hoist supplied with broken rock for several months.

The amount of drilling and of powder used are so regulated that as few large blocks as possible will be brought



Figure 11.—Thetford Mines, Quebec.

Face of asbestos pit, with cable-way towers, mill, and dumps; barren rock in background.

down in the blast. Blocks too large to be easily handled must be drilled by means of plugger drills and again blasted. The broken rock is sorted and loaded into boxes by shovellers, barren rock going into one set of boxes, and rock containing fibre for the mill into another set. All crude fibre broken loose in blasting is picked up and taken to the cobbing shed, where it is freed from adhering particles of rock and graded into No. 1 and No. 2 'crude', according to length of fibre.

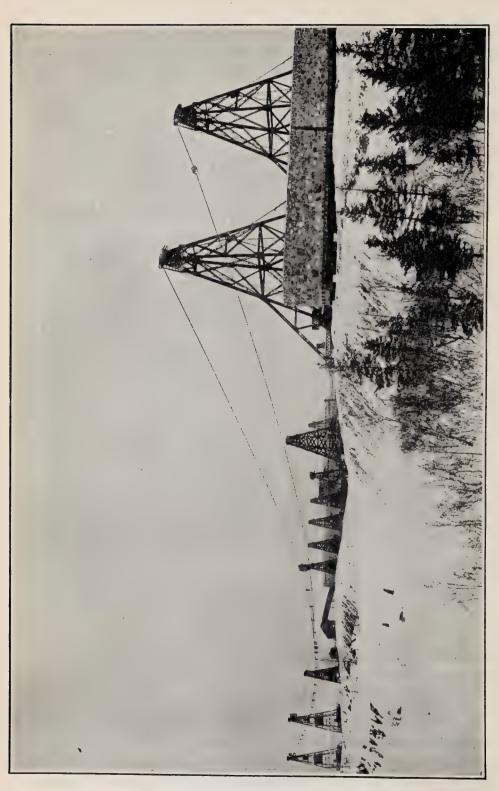
At one mine the drilling method is varied, and vertical holes are drilled from the top of the face by means of ordinary hammer drills and long steel—well-drilling methods, as employed in stone quarries, have not been adopted in the district. At this mine, the loaded boxes are lifted by steam



Figure 12.—East Broughton, Quebec. Loading asbestos rock by steam-shovel.

cranes and dumped into cars which are hauled up an inclined track to the mill. Ore which does not require sorting is loaded by steam shovel into the cars. The use of the crane or steam shovel, where feasible, is more economical than the cable derrick.

At another property an underground mining system combined with glory-holing has been installed. A regular system of shrinkage stopes provides underground storage for a supply of ore. This system has been found to have several disabilities peculiar to the area.



In the westerly, or Black Lake, part of the field, glory-holing has been successful in providing a low-cost mill feed. Where crude fibre occurs, underground or glory-hole mining, or steam shovel loading, cannot be expected to produce maximum results unless supplemented by picking belts on which the crude fibre may be recovered.



Figure 14.—Cable-way tower, with cable anchor in foreground.

CABLE-WAY HOIST

As the cable-way hoist has been developed in the Thetford district of Quebec in larger units than elsewhere, and as the cable-way is so generally used in deep open pit work, a

description of the most recent large installation is given here. This is supplied by Mr. H. V. Haight, designing engineer of the Canadian Ingersoll-Rand Company. The cable-ways were constructed at the instance of the Asbestos Corporation, the officials responsible for the installation of a pair of cable-way hoists of this size being the manager of the Corporation, Mr. R. P. Doucet, and the chief engineer, Mr. C. V. Smith.

"In 1914, Asbestos Corporation of Canada, Limited, now Asbestos Corporation, Limited, installed at their King mine, Thetford Mines, Quebec, a moveable cable-way of 937 feet span handling a rock load of six tons. This replaced smaller equipment then in use. Additional cable-ways of the same capacity were later erected, till they now have six of this size.

"In order to supply their new mill, completed in 1924, the pits are being deepened at such a rate that it was decided to widen them by installing two larger cableways. These new cableways have spans of 1,400 feet and lift a net rock load of ten tons. As the remainder of the pit is widened by these new cableways, the old ones of shorter span will be advanced and new ones of the larger span erected in their place. The new ones also operate at increased speed, as shown by the following table:

	Old Cableways	New Cableways
Span, centre to centre of towers	937 feet	1,400 feet
Maximum depth of pit from surface.	400 feet	700 feet
Capacity of ore box	132 cu. ft.	220 cu. ft.
Net weight of ore	12,000 lb.	20,000 lb.
Weight of ore box	2,700 lb.	4,400 lb.
Total suspended load at centre (in-		,
cluding sheaves, ropes, and car-		
riage)	22,000 lb.	36,000 lb.
Rope speed (trolleying speed)	1,000 ft.per min.	1,500 ft. per min.
Hoisting speed (3-part line)	333 ft. per min.	500 ft. per min.
Horse power of hoist motor	200 h.p.	450 h.p.
Diameter of hoisting ropes	3/4 in.	7/8 in.
Diameter of main cable	$2\frac{1}{2}$ in.	3 in.
Stress in main cable	126,000 lb.	240,000 lb.

"Figure 13 shows the general arrangement of the new cableways. Figure 14 shows a 'close-up' of one of the travelling head towers, with its counterweight of rubble masonry, and also the travelling ore bins, a cableway carriage, an ore box just about to be dumped, and the fall-rope carriers with their spacing chain. Other views of the towers are shown in figures 15 and 16.



Figure 15.—Cable-way towers, 105 feet high.

"The general method of operating the cableway requires very little explanation. The carriage is pulled back and forth on the main cables by an endless rope on the rear drum.

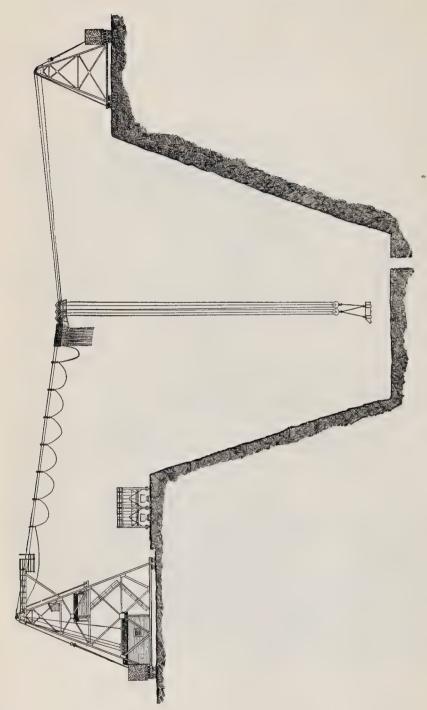


Figure 16—Cable-way hoist at Asbestos Corporation Pit, Thetford Mines, Que.

The hoisting is done by the centre drum. The front drum carries the dumping rope, which does very little work except at the instant of dumping. The hoisting and dumping drums usually turn together, but when the box is to be dumped, the brake is set and the clutch released on the hoisting drum and then the dumping drum lifts the rear end of the ore box, letting the ore slide out the open end.

"The hoist has certain special features that merit brief description. It is located in the base of the tower, but is arranged for remote control from the operator's cabin,

higher up.

"The motor is 450 h.p., 600 r.p.m., 2,200 volts, 3 phase, 30 cycles. It has full magnetic control, and all the electrical

equipment is in the upper cabin, near the operator.

"Each drum of the hoist has a cone clutch and a band brake. In addition, there is a main service brake on the pinion shaft. It is a graduated brake of the gravity-air type, set by weights and released by an air cylinder, the latter controlled by floating gear which is connected by rods to the operator's stand. The operator thus has instant, accurate, and easy control of the braking action. The brake blocks are of asbestos and the brake is very powerful, the brake drum being 24 inches in diameter with brake blocks 16½ inches wide.

"The three-band brakes on the drums are also set by weights and released by air, but are of the 'off and on' type,

controlled by valves at the operator's stand.

"The clutches are of the cone type. The clutch is set or released by a pair of toggles and a sliding sleeve, the sleeve being operated by an air cylinder, which is controlled by valves at the operator's stand. This type of clutch has many good features. Like the Lane band clutch, it is locked when engaged and has then no unbalanced end thrust on the bearings. It is, however, superior to the band clutch in several respects: it is in perfect running balance; it has no tendency to drag when released; the take-up for wear is very convenient; and it is very safe, the main parts being in compression.

"The construction of the clutch on the endless rope drum is identical with that just described, but the drum itself is slightly different. After travelling across the face of the drum, the rope climbs the fillets in the corners of the drum and then crowds over, as on a spool. The fillets may be replaced when worn. Experience has shown that 5½ turns around the endless rope drum gives sufficient friction.

"The dumping drum was at first made with a togglecone clutch similar to those on the other two drums. This. however, did not work properly. Owing to the rather large fleet angle and the low strain on the dumping rope, it would sometimes climb to the next layer before reaching the end of the drum. The increased diameter then made the dumping rope wind faster than the hoisting rope, and it started to dump. To remedy this, the toggles were taken off and a screw thrust substituted. The thrust and counter-thrust are taken by bronze washers running between hardened steel plates. The bronze plates are grooved. These thrusts run in oil. The arm on the thrust screw is connected by rods to a hand lever at the operator's stand. With this arrangement, the operator sets the clutch on the dumping drum very lightly, and if the rope should start to climb the clutch will slip. When dumping, the operator can set the clutch harder if necessary.

"The gears are all Falk cut steel herring-bone gears, and are enclosed in an oil-tight sheet steel gear case.

"The whole hoist is mounted on a very substantial sectional cast iron bed, the members being of box pattern. The motor is carried on an extension bolted to the main bed. The motor frame is of special design, having the feet nearer the centre-line than usual, in order to make the bottom of the bed under the motor of the same height as the main bed. This makes it convenient to attach the hoist to the base framing of the tower.

"The general engineering of the cable-ways was handled by the staff of Asbestos Corporation, Limited. The span, the loads, and the rope speeds are unusually high.

"The structural steel work was designed by Dominion Bridge Company; the main cables and hoisting ropes were made by Dominion Wire Rope Company, Limited, of Montreal; the electrical equipment by Canadian General Electric Company, Limited; and the hoists were designed and built by Canadian Ingersoll-Rand Company, Limited."

On the westerly extension of the field, at Asbestos, near Danville, a large open pit is worked in benches by



Figure 17.—Cable-way tower, rock bins, carriage, and carriers.

steam shovels in a manner similar to that employed on the iron ranges of Minnesota, in the porphyry copper mines of the western United States, and at Rio Tinto, Spain.

MILLING PRACTICE

While the mining methods employed are merely variations of those in general use in getting out any rock, asbestos milling has developed some features of its own. It is becoming more generally recognized by the operators that the rock in each section requires to have a mill specially designed for its individual treatment. The hardness of the rock varies from place to place, as also does the quality, length, and percentage of the fibre. A mill which will recover fibre efficiently in one area may be so harsh in its

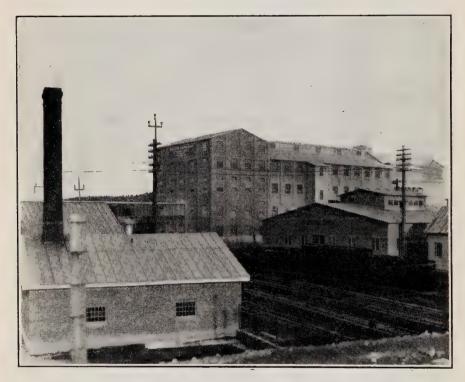


Figure 18.—Thetford Mines, Quebec.
An asbestos mill.

treatment as to be destructive of length of fibre in another. A soft rock which will produce only short or low-grade fibre in big tonnage must be milled in a different manner from a hard rock containing a comparatively small amount of fibre but that of good quality and length.

It has taken considerable experience and the building of many mills to demonstrate this fact. The present tendency in mill design is to eliminate unnecessary machinery by lifting fibre after each crushing, unloading sand as soon as made, and keeping hard barren rock out of the mill.

Other than the cyclone, three varieties of which have been designed and developed in the Quebec district, all units in any asbestos mill are such as are used in other dry milling metallurgy.

One of the most useful advances has been the building of large bins for the storage of the crushed rock after drying. This enables the mill-man to draw feed containing an average fibre content, so that the mill will not be loaded with fibre one day and be running on lean rock the next. Moreover, the rock in the bin cures to an even moisture content, so that hasty adjustments need not be made to accommodate a dry dusty fibre one hour and a damp tough fibre the next.

The necessity of ample screening area in the mill is being more generally recognized, although, with the type of flat shaking screen commonly in use, the screen cloth is cleaned only with difficulty and is nearly always blinded

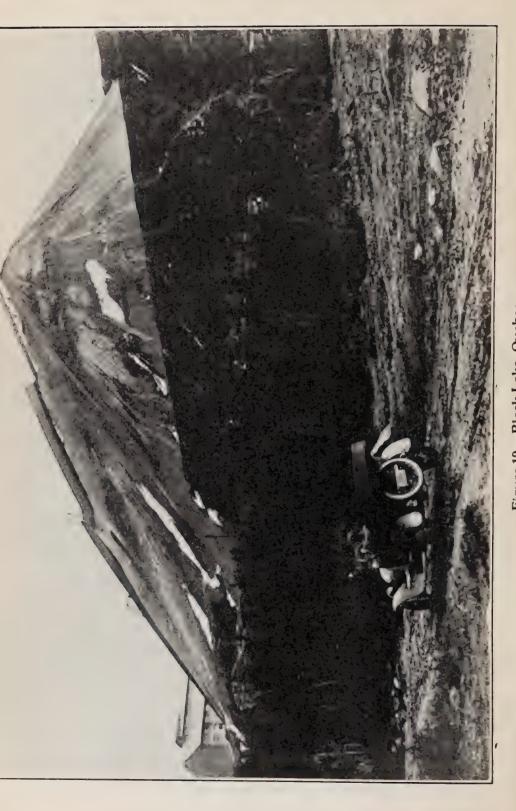
and ineffective for the larger part of its surface.

Vibrating screens, both electrical and mechanical, are being found useful for removing sand, but they must be followed by flat shaking screens on which the rock and fibre will bed and from which the fibre may be lifted by suction. More attention is being given to cleaning the fibre thoroughly, and each year less dust is left in the fibre by the better mills. The very fine fibre is collected in large dust bins and sold as 'floats'. In newer mills, fibre and sand are not allowed to pass through the suction fans, nor is opened fibre allowed to go into a crusher.

With the control of the properties now in the hands of a small group of operators, instead of each property being operated as a separate unit as in the early days of the industry, the number of grades of fibre produced and marketed has been reduced from fifty to about a dozen.

The design of a mill, in most instances, depends on the personal predilection of the operator of the moment, or on the persuasive ability of machinery salesmen.

In order to reduce costs, increase percentage of fibre recovered, and improve grades, advantage has been taken, in some mills recently built, of the experience of the consulting



engineer, and successful practice from other fields has been introduced. Notable improvements in milling and reductions in costs have resulted from such co-operation.

The competition of operators to mill a large tonnage of rock causes much barren rock to be milled. This is ground into sand at a higher cost than that at which it could be picked out and discarded in lump form. The milling of such material also reduces the capacity of the mill, wears the machinery, and cuts up the fibre.

It too often happens that fibre, after being freed from rock, is allowed to enter the next crushing unit along with sand which should have been eliminated, and as a result it is unduly cut up.

The structural geology of the area has not been worked out in detail, and at times mills are running on low-grade rock because the high-grade zones have not been delimited and it is more convenient to mine the low-grade.

With a fuller knowledge of the content of their ground, operators would be in a better position to produce the grades of fibre demanded by the market at any time, and secure a larger tonnage of fibre from the existing milling equipment.

A few typical flow-sheets will indicate the diversity of ideas for milling similar rock.

Mill B

Mill A
Jaw crusher
Rolls
Hammer crusher
Dryer
Storage bin
Screens with suction
Hammer mill
Screens with suction
Cyclone
Screens with suction
Jumbo
Screens with suction
Jumbo for tailings
Screens with suction
Rotary graders and clear
ing screens

Floats bin

Jaw crusher Jaw crusher Grizzly screen Picking belts Dryer Disc crusher Screens with suction Jumbo Screens with suction Jumbo Screens with suction Grading and cleaning

Mill C Jaw crusher Dryer Storage Jaw crusher Hammer crusher Screens with suction Hammer crusher Screens with suction Cyclone, tailings Screens with suction Dust bin

Mill D	Mill E	Mill F
Dryer	Grizzly	Jaw crusher
Jaw crusher	Jaw crusher	Dryer
Jaw crusher	Jaw crusher	Storage bin
Rolls	Trommel screen	Hammer mi
Rolls	Dryer	Screens with
Jumbo	Disc crusher	Cyclone
Screens with suction	Screens with suction	. Screens with
Jumbo	Jumbo	Grading and
Screens with suction	Screens with suction	screens
Jumbo	Rotary graders and	
Screens with suction	cleaning screens	
Jumbo	Ball mill for tailings	
Screens with suction		
Jumbo		
Screens with suction		
Jumbo		
Screens with suction		
Jumbo		
Screens with suction		
Flat grading screens		

rill. h suction

h suction d cleaning

WET MILLING

In all existing mills the rock is dried before going to the mill, and is milled in the dry state. Interesting experiments on rock from various pits have been carried on during the past few years, based on the theory that by crushing the rock in water the fibre will be freed and float off, after which it may be dried, fiberized, and graded by the usual methods.

It is claimed that a larger percentage of fibre is thus recovered, and more of it in the longer grades. Large-scale test runs are about to be made, and the advantages, or disadvantages, of the new system will shortly be demonstrated in a commercial plant.

PRODUCTION

With the merging of seven properties in 1926, the number of companies engaged in the Ouebec asbestos field has been reduced to five, who operate a total of eleven pits and fourteen mills. Eight mills formerly operated are at present inactive.

The accompanying tables give the production for the years 1925 and 1926, as reported by the Department of Mines for the Province of Ouebec.

Production of Asbestos in the Province of Quebec

1926

Designation of Grade	Š	Sales	Average Value	Stocks Dec. 3	Stocks on hand Dec. 31st, 1926*
)	Tons	Value	per ton	Tons	Value
Cude No. 1	1,094	\$ 406,438	\$371.51	606	\$ 337,703
Criide No. 2	3,494	802,304	259.62	145	33,295
Cride rin-of-mine	446	92,394	207.16	297	61,527
Spinning fibre	14,482	1,885,835	130.22	2,115	275,415
Shinole fibre	36,497	2,139,780	58.62	2,138	125,330
Mill board and naner fibres	86,746	2,940,675	33.88	29,499	999,426
Fillers, floats and other short fibres.	135,930	1,828,061	13.45	21,038	282,961
Totals	278,689	10,095,487	36.22	56,141	2,115,657
By-products (sand and gravel)	15,672	10,257	0.65	•	•
Totals	294,361	10,105,744		56,141	2,115,657
		_			

Quantity of rock mined during the year 1926: 4,479,138 tons.

^{*}Values calculated at average price of each grade. These figures are given merely as an approximate guide of valuation of stocks on hand.

Production of Asbestos in the Province of Quebec

1925

Designation of Grade	Š	Sales	Average Value	Stocks Dec. 3	Stocks on hand Dec. 31st, 1925*
	Tons	Value	per ton	Tons	Value
Crude No. 1 Crude No. 2 Crude run-of-mine. Spinning fibre. Shingle fibre. Mill board and paper fibres. Fillers, floats and other short fibres.	1,044 3,777 348 16,070 30,010 93,935 128,338	\$ 381,025 778,895 49,030 1,710,379 1,523,980 2,915,046 1,618,290 8,976,645	\$364.96 206.22 140.90 106.43 50.78 31.03 12.61	1,191 704 340 3,115 4,536 8,663 16,623	\$ 434,667 145,179 47,767 331,529 230,338 268,813 209,616 1,667,909
By-products (sand and gravel)	16,865	10,814	0.64		
Totals	290,387	8,987,459		35,172	1,667,909

Quantity of rock mined during the year 1925: 4,121,258 tons.

*Values calculated at average price of each grade. These figures are given merely as an approximate guide of valuation of stocks on hand. The following table indicates the comparative production from various countries, as unmanufactured asbestos, in long tons:

		4005
Country	1924	1925
Country	208,762	273,522
Canada		34.349
South Africa:—Rhodesia	26,141	
Union of South Africa	6,464	9,078
	3,940	8,000
Cyprus	8,331	
Russia		
Italy	2,126	2,490
T 1	125	(not available)
India		(not available)
Australia		
	239	330
China	268	1.123
United States	200	1,120

In Canada, the present producing companies are: The Asbestos Corporation, operating five pits and five mills, with a sixth mill undergoing reconstruction; Johnson's Company, with two pits and two mills in operation; Keasbey, Mattison Company, operating one pit and one mill; Canadian Johns-Manville Company, Limited, operating one pit and four mills; and The Quebec Asbestos Company, operating two pits and two mills.

In Rhodesia, the properties producing are: In the Bulawayo District, the Nil Desperandum (African Asbestos Mining Company, Limited); Pangani (J. S. Hancock); and Shabani (Rhodesian and General Asbestos Corporation, Limited). In the Victoria District, Gath's (Rhodesian and General Asbestos Corporation, Limited); and the King, of

the same Company.

Prior to 1910, the total amount of asbestos produced in South Africa was 327 tons, so that the present rate has been attained in the short space of 16 years. As recently as 1915 the production for the year was less than 2,000 tons. With the exception of 1922, the production has shown a continual

increase year by year.

The average fibre content of the rock milled in Quebec during 1926 was 6.02 per cent. In January, 1927, one pit ran on a large tonnage of rock containing 16 per cent of high-grade fibre. The lowest grade of rock milled profitably at any time contained 1.5 per cent of fibre, of which a large percentage was recovered in the longer grades. This operation was successful, however, only during a short era of unusually high prices for asbestos.

In 1923 some interesting statistics on production in Canada were compiled by the late George L. Burland. Brought up to

date, these are reproduced in the accompanying tables.

CANADIAN PRODUCTION DATA

Rock mined and milled, and value per ton of rock

% Fibre per ton rock	6.20 6.20 6.20 6.20 6.20 6.20 6.20 6.20	20.02
Rock milled (tons)	1,477,613 1,571,310 2,123,024 1,808,285 1,813,961 1,947,424 2,239,249 2,078,883 2,502,436	•
Average value per ton rock	1.59 1.48 1.49 1.50 1.51 1.51 1.51 1.51 1.51 1.51 1.51	ì
Asbestos per ton rock (pounds)	107.8 108.6 105.4 111.4 96.9 102.6 100.8 100.8 107.2 107.2 107.2 107.2 117.0 117.0 117.0	
Rock mined (tons)	1,583,076 1,870,608 2,527,410 2,127,395 2,231,087 2,634,410 2,445,745 3,061,690 3,123,370 2,224,138 2,920,280 3,747,576 4,121,258 4,479,138	
Average value per ton	29.60 27.52 28.04 31.33 38.87 52.45 63.35 63.35 63.35 80.35 31.37 32.82 36.22	_
Total value shipments (dollars)	3,026,306 3,059,084 3,830,504 2,895,935 3,544,362 5,188,598 10,932,289 10,932,289 14,749,048 5,199,789 6,053,068 7,364,260 6,561,659 8,976,645 10,095,487	
Fibre shipped (tons)	102,224 111,175 136,609 107,401 113,115 233,339 137,242 142,375 135,862 179,891 87,475 160,339 216,804 208,762 273,322 273,322	_
Year	1911 1912 1913 1914 1915 1916 1919 1920 1921 1922 1923 1924 1925	-

CANADIAN PRODUCTION DATA (Continued)

Percentage value of sales to total sales

Short fibres only, combined	60.8 63.0 53.0 53.0 53.0 647.7 47.7 47.7 63.6 63.6 63.6 63.6 63.6 63.6
Total mill stocks	74.6 74.6 71.0 74.6 69.8 62.6 61.8 61.8 70.4 74.2 88.5 88.5 88.5 88.1
Crudes and spinning, combined	339.2 55.55 55.55 33.35.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55 33.55
No. 1 and No. 2 crudes, combined	25.4 29.0 29.0 26.8 37.2 38.4 38.4 27.8 29.6 11.0 12.9 11.9
Total crude and mill stocks	%8899999999999999999999999999999999999
Year	1911 1912 1913 1914 1915 1916 1917 1920 1921 1922 1923 1924 1925

EXPORTS OF ASBESTOS

Exports of asbestos fibre, and of asbestos sand and waste, from Canada in 1926 are reported to the following countries, in the amounts and values noted:

Country	Asbestos fibre			Asbestos sand and waste	
	Tons	Value	Tons	Value	
United Kingdom United States Australia Belgium Denmark France Germany Italy Japan Mexico Netherlands Other Countries	7,710 92,897 1,605 10,033 126 6,860 12,537 3,671 4,518 80 1,723	\$ 575,866 5,295,168 116,250 628,981 6,600 481,145 900,104 242,482 250,714 5,450 157,050	1,579 120,781 10,961 351 1,656 123 720 60	\$ 35,467 1,743,635 151,168 7,517 34,718 2,775 15,850 1,350	
Total	141,760	\$8,669,810	136,231	\$1,992,480	

Grand total: 277,991 tons, value \$10,662,290.

The average value of fibre shipped from Rhodesia in 1926 is reported as \$115.00 per ton.

The total African exports for 1926 are estimated at 34,000 tons valued at \$5,500,000. The Rhodesian output alone has increased from 487 tons in 1914 to an estimated total of 38,000 tons, valued at \$4,500,000, in 1926.

MANUFACTURING

In Canada, only three companies manufacture fibre into finished products, such as brake lining and shingles.

In the United States, in 1925, forty-nine establishments were engaged primarily in the manufacture of asbestos products, not including steam packing or boiler or pipe covering. The total output of manufactured asbestos goods was valued, in 1925, at \$36,173,797, made up as follows:

Brake linings and clutch facings.

Shingles and roofing materials.

Table mats and protectors.

All other products.

Detailed reports on the progress of the industry in Canada may be found in the annual reports on Mining Operations in the Province of Quebec, issued by the Quebec Department of Mines. Some of the illustrations accompanying this article were supplied by the Department.

DEWATERING THE LOWER LEVELS OF THE SIMMER AND JACK MINES, LIMITED*

By R. Craib (Member, S. Af. Inst. Eng.) **

(Vancouver, B.C., Meeting, September 14th, 1927)

The nature and magnitude of the task of dewatering the lower levels of the Simmer and Jack will perhaps be better understood if their extent, their correlation to adjacent mines, and the circumstances which led to their being flooded, are

first briefly considered.

The Simmer and Jack Proprietary Mines, Limited, now incorporated in the Simmer and Jack Mines, Limited, was commenced on outcrop reefs dipping south at an average angle of 35 degrees to the horizontal, the southern boundary of the mine being some 4,500 feet from the outcrop. The Simmer Deep abutted on this southern boundary; on the east were Rose Deep and Knights Deep, and on the west the Geldenhuis Deep and Jupiter G. M. Co., Ltd. (see key plan, Figure 1). There is through connection underground between all these companies.

The Simmer Deep sank vertical shafts approximately 350 feet south of the Simmer and Jack boundary to a depth of 3,116 feet, from which incline shafts were continued to an

ultimate depth of 4,822 feet vertical.

During 1914, a central pumping plant was installed at the bottom of the Lohse shaft, Knights Deep, for the purpose of raising to the surface, in addition to its own drainage, certain water from the Rose Deep, Simmer Deep, Simmer and Jack Proprietary Mines, Ltd., and Jupiter, the ultimate aim being that this central pumping plant should handle the whole of the drainage from these mines. The primary idea was to effect a saving in pumping costs by the elimination of smaller

**Resident Engineer, Simmer and Jack Mines, Limited, Transvaal, South Africa. Past President, S. Af. Inst. Eng.

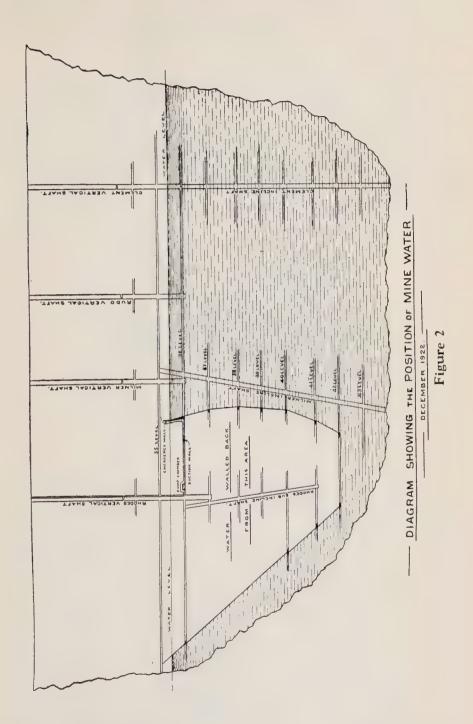
^{*}This paper was published in the Journal of the S. Af. Inst. Eng., Vol. XXV, No. 4, 1926.

and less efficient individual units. Mining operations had not sufficiently advanced in 1914 to permit of the application of the scheme in its entirety, and the Simmer and Jack had still to retain and use most of its own pumping plant, which consisted mainly of Cornish, air and belt-driven reciprocating pumps, but subsequently practically all its water was delivered to the central pumping plant.

In 1920 two unfortunate occurrences — the closing down of the Simmer Deep and the Knights Deep — gave rise to an entirely new state of affairs. The Simmer Deep, mainly because of post-war conditions, the most serious by far being white labour troubles, had finally to cease working in July of that year. The workings were stripped of their rails and other equipment, for which there was a good market, and allowed to flood. Owing to a disastrous fire, which completely destroyed the mill, Knights Deep closed down in September, 1920, so that the arrangement for raising the Simmer and Jack drainage through the Lohse shaft ceased. Its water was then allowed to gravitate to the Simmer Deep, into which also flowed water from the adjacent mines.

It was estimated that the Simmer Deep would flood in approximately three years, by which time the Simmer and Jack was expected to be worked out. Towards the end of 1921, however, the level of the water had risen to a height which made it imperative for the Simmer and Jack to take precautionary measures against the flooding of the forty claims which it had acquired from the Simmer Deep in 1917, and which were in jeopardy from the rising water. The means adopted were the building of concrete walls which dammed back the water on both sides of this block of claims and permitted mining operations to be carried on at a depth of 922 feet below the surface of the water. Figure 2 shows in schematic arrangement the water level relative to the bottom workings towards the end of 1922.

While these walls were in course of construction and the level of the water gradually rising, an agreement was entered into between the Simmer and Jack and Rose Deep for the installation of a joint pumping plant at the foot of the Rhodes vertical shaft, the purpose being to prevent the water rising



above a point midway between 36 and 35 levels. The overflow at this point was to be raised to the surface through the Rhodes shaft by means of centrifugal pumps having a manometric head of 3,256 feet.

From the bottom of the Rhodes subsidiary incline the drainage was raised to the 35 level by means of stage pumping, and from there allowed to gravitate, along with the drainage from higher levels, to the top of the Milner incline. By this arrangement the whole of the Simmer Deep area served as a settling sump and clear water was delivered to the joint pumping plant. It may be remarked here that, as the water entered the Milner incline, lime was added to neutralise the acidity.

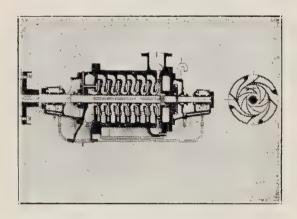


Figure 3

The joint pumping plant consists of a high-lift centrifugal pump of the well-known Sulzer design, with high and low pressure units having eight and seven stages respectively, being driven by a South African General Electric Company's 2,000 h.p. motor, placed centrally (see Figure 3). The pumps are of simple design, but are characterized by accuracy in machining. They are built with a cast steel outer casing having suitable ports on the delivery end which lead into the delivery column. The casing is lined with a quarter inch bronze protection liner. Each of the stages consists of an impeller, guide wheel, overflow piece and overflow protection plate. The front or inlet cover is made of bronze, while the rear cover is of steel with bronze protection plate. With the exception of the impellers, the whole assembly is pressed tightly together by

means of the rear cover. The impellers are held in position on the shaft by two long feathers, while a threaded bronze nut presses the boss of one impeller against the other, thereby forming one complete bronze sleeve over the steel shaft.

On the outer side of the rear cover is placed a balancing disc which takes the place of a thrust bearing. This disc, in conjunction with its wearing sleeves, is probably the most important part of the whole pump. The end thrust on a multi-stage pump is greater than one would appreciate, and on the one under consideration would probably be in the neighbourhood of 50,000 lb. when working under normal conditions. The outer bush to the balancing disc is fastened to the rear cover by means of set screws, whilst the balancing or revolving disc is screwed on to the shaft and holds in position the wearing sleeve.

The action of the balancing disc is as follows. Between the last impeller and the rear cover there is a water pressure equal to the delivery head which forces a certain amount of water into the space between the revolving and the stationary discs, according to the clearance between the outer bush to the balancing disc and the rear spacing bush. This pressure builds up until it eventually overcomes the end thrust, at which time the revolving disc then moves away from the stationary disc and allows the water to escape into the balancing disc chamber. As the pressure decreases, the revolving disc comes closer to the stationary disc, when the pressure again rises and so pushes the revolving disc away. In this way a constant movement is maintained, measured probably in thousandths of an inch, and can be likened to a sensitive governor for the purpose of equilibrium.

It is obvious, therefore, that any increase in clearance between the rear spacing bush and outer bush to balancing disc must of necessity decrease the efficiency of the pump. Working under normal conditions, with a clearance of fifteen thousandths, the pump will pass 18 per cent of the total quantity of water delivered through its balancing disc. This, however, must not be considered a drop in efficiency, as the makers have taken it into consideration and made allowance in their pump rating. It is only when the clearance between

these bushes becomes greater than fifteen thousandths and the discharge increases in proportion that the general efficiency of the pump may be said to decrease. The pump delivers 75,000 gallons per hour against a manometric head of 3,256 feet, which is quite exceptional and probably the highest lift of any pump in the world.

On the delivery side of the pump is a cast steel retaining valve to which is attached a cast steel gate valve suitably geared to permit of its being opened and closed by hand.

Between the pump and the vertical column are two 24-ft. lengths of 10-in. Mannesmann weldless steel tubing, the walls of which are ½-in. thick, bolted together by the 'double border' type of joint. At the foot of the vertical shaft there is a cast steel duck-foot bend resting on a substantial block of concrete designed to carry the first thousand feet of column plus the weight due to total head. The bottom portion of the delivery column consists of 432 feet of Stewarts and Lloyds piping, having a thickness of ½-in. The centre portion is made up of 1,200 feet of Mannesmann piping ¾-in. thickness, while the top or last length of 1,416 feet is reduced to ¼-in.

The column is supported in the shaft by means of 6-in. by 8-in. pitch pine bearers, placed centrally every second length (see Figure 4). Attached to the pipes and resting on these bearers are 11-in. clamps made from 6-in. by 1-in. section mild steel. On the long end of the clamp is welded a lug to prevent the pipes disengaging from the bearer. In addition to these timber supports, the column is also supported every thousand feet by means of two 24-in. by 7-in. girders, the inside girder being hitched and concreted into the side of the shaft, while the outside girder rests on a shelf which permits of its removal if required. The column is secured to these girders by six 12-in. by 1-in. clamps having six 11/8-in. bolts each. The bottom clamp, with suitable footstools, rests on two 18-in. by 6-in. girders. This method gives sufficient head room for two fifty-ton hydraulic jacks to be placed in position to raise the column if required. There are no expansion joints installed and, during the three and a half years which the column has been in use, there have been no indications that such joints are necessary. The column is carried one length higher than the collar of the shaft, but has a cast iron tee immediately below the collar set from which a delivery pipe is led to the recording weir.

The weir is a very simple design, being made of 9-in. by 3-in. T. & G. boarding, and is illustrated in Figure 5. The internal dimensions are 15 ft. by 5 ft. by 4 ft. deep. There is a baffle board 24 in. from the inlet end which drops to within 12 in. of the bottom. This baffle steadies the running water and creates a smooth surface on the overflow end. The box is bolted together and suitably braced with 6-in. by 3-in. timbers. At the overflow end is a 'V' notch 21 in. deep, made from ½-in. mild steel plate bevelled on the down stream edge. The clockwork for operating the registering drum was obtained from an old Venturi meter which, with certain alterations, has proved entirely satisfactory. Immediately under the drum is placed a 15-in. by $2\frac{1}{2}$ -in. glass tube which is connected to the side of the measuring weir by means of a 1-in. pipe. The

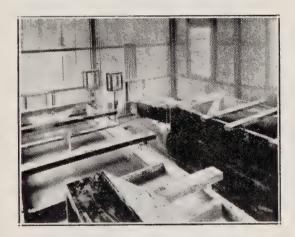
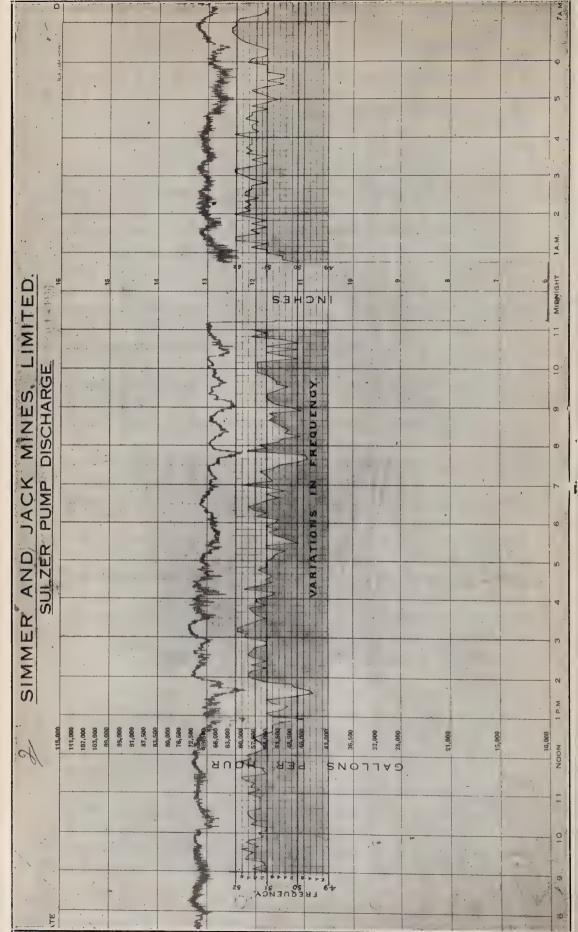


Figure 5

water in the glass therefore rises and falls according to the level of the water in the weir. The actual fluctuation of the level of the water is registered on a printed chart which is marked for both inches in height and gallons per hour by means of a pen attached to the top of an aluminium rod carried on a copper float. Figure 6 is a copy of one of these charts. The fluctuations are caused by variations in the frequency of the power supply. An efficient measuring weir is essential as it registers immediately any drop in the efficiency of the pump.



The power for operating the pump is transmitted from a transformer station on the surface to a distributing panel in the pump chamber by 0.2 sq. in. cables carrying 2,000 volts. The cables are clamped to the shaft timbers every twelve feet, while junction box compartments are suitably placed for thousand feet lengths of cable.

Owing to worked out areas and the general condition of the drive, the suction layout is exceptional. On 36 level and 1,350 feet from the pump chamber an eight-foot concrete inlet wall was built, and in this are placed two 14-in. internal diameter cast iron pipes, 1½-in. thick, cast iron being chosen on account of its being less subject to corrosion. A 14-in. cast iron gate valve is bolted on each pipe on the pump side of the wall, from which projects a short length of wrought iron pipe having attached to it a suitably proportioned mild steel strainer. In the strainer are placed two screens composed of 1-in. and ½-in. meshed galvanized iron wire netting to prevent any foreign material entering the suction pipe. Figure 7 shows the strainer with the arrangement for cleaning.

From the strainer to the pump the suction pipe consists of 14-in. internal diameter wrought iron pipes, in 18-ft. lengths where possible, which are joined by means of the well-known Kimberley lead and socket joints. An objectionable feature in the suction column is the number of bends and joints in consequence of the irregular course of the drive. As will be seen presently, however, this pipe line was seldom called upon to carry a suction head and so far the suction lay-out has functioned satisfactorily. The ideal condition, of course, would have been to have built the inlet wall close to the pump.

Another important feature in connection with the suction lay-out calls for explanation. An emergency device had to be arranged to ensure safety against fracture of either the valve or pipe on the intake wall. This might have been effected by inserting a bronze valve on the intake side of the wall, operated on the level above by means of gear wheels and shafting, but there would have always been the danger of such appliance failing to operate when urgently required. It was therefore decided to build, in a disused stope, a concrete wall with a series of water-tight doors, which could be removed

or placed in position as occasion required. Figure 8 shows the construction of the wall, cast iron doors being placed at 3 ft. 6 in. centres. It will be observed that the doors are designed for external pressure, which was all that was necessary at this time. During a portion of the dewatering period, however, the lowest door was called upon to withstand an internal pressure of 25 lb. to the square inch, or a total pressure of five tons.

The Rose Deep and Simmer and Jack joint pumping plant was put into commission in November, 1922, and for the following twenty-three months this single unit handled the whole of the underground water and kept it back to about 35 level. There was little fear of flooding in case of breakdowns as, by increasing the height of the emergency wall, a storage capacity of thirty days could be obtained — quite a reasonable time in which to effect repairs. As a matter of fact, on one occasion, pumping was stopped for fifteen days.

In October, 1923, the Simmer and Jack Proprietary Mines Ltd. leased from the Government the Simmer Deep claims. The augmented property was then named Simmer and Jack Mines Ltd. The claims were entirely submerged and had to be dewatered before mining operations could be resumed. This was a matter of urgency as it was imperative that the Simmer and Jack should be in a position to work the newly acquired ground at the earliest possible moment, because its original area was becoming rapidly exhausted.

An estimate was made of the quantity of water that had to be removed, by multiplying the number of months it had taken the mine to flood by the average monthly quantity pumped to the surface after the mine had flooded. This gave a figure in the neighbourhood of 500,000,000 gallons. From the underground plans, the surveyors calculated the quantity to be 400,000,000 gallons. This latter figure was accepted, so that the problem that presented itself was the raising to the surface of some 400,000,000 gallons of accumulated water, as well as the regular mine flow of some 700,000 gallons every twenty-four hours.

Dewatering was done through the Milner incline and, in the investigation as to the most suitable and effective plant for accomplishing this work, the dimensions of the shaft and the voltage of electric power supply had to be taken into consideration, these two factors determining the size of plant that could be installed.

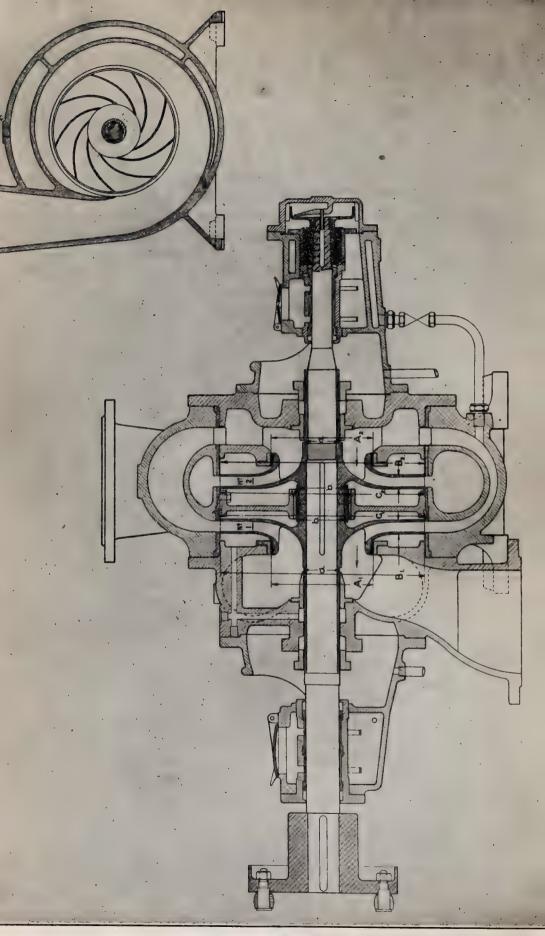
It was known that the shaft measured, untimbered, 26-ft. by 7-ft., but these dimensions could not be accepted since, at the closing down of the Simmer Deep, the shaft had not been entirely stripped. A drawing of the skips which had served the shaft provided the only reliable data for head room. The height of the skip was 4 ft. 6 in. which, with two inches clearance above, gave a maximum height of 4 ft. 8 in. for the passage of a plant. The width of the shaft sufficed for practically any type of plant.

Electric power of 2,000 volts was adopted, this being the standard voltage in use on the mine and, moreover, taking into consideration the rough and wet nature of the work to be performed, this voltage was deemed to be the highest that could be used with comparative safety.

Information concerning the removal of large bodies of water at great depths was meagre. After careful consideration of such and having regard to the imperative need of speed, it was decided to instal centrifugal pumps and motors, as far as practicable, of standard designs, an essential feature, however, being that the pumps should be built entirely of acid resisting material.

With a view to obviating stoppages from breakdowns, and as a safeguard in the event of sudden influx of water, it was decided to instal two complete sets of two-stage Sulzer standard horizontal medium-lift centrifugal pumps, each direct coupled to a 450 h.p. South African General Electric Co. motor, one left and one right hand, with suction and delivery branches 38 degrees to the horizontal. One set was to serve as a stand-by. Each pump has a capacity of 100,000 gallons per hour against a manometric head of 500 feet when running at 1,470 r.p.m. These sets are hereafter referred to as the sinking plant.

Figure 9 shows a section of one of the sinking pumps. It will be observed that there are no guide vanes and, in view of the grit and sludge which the pumps would have to handle,



any increase in efficiency by the use of guide vanes would have been counteracted by the increased wear and tear due to them.

The casing of the thrust bearing is made of cast iron, lined with white metal, in which runs a collar having ten grooves. The total bearing surface is 44.18 sq. in., and is lubricated from an oil well by means of a scoop attached to the end of the shaft. The whole thrust is water-cooled by a jet of water from the first stage chamber.

Although provision is made for an abnormal end thrust towards the motor, it will be found by calculating the various thrusts that the out of balance load, provided the pump is in perfect condition, should not be more than, say, 1,083 lb., as appears from the following:

Let diameter a= an area of 8.946 sq. in., diameter b= an area of 19.019 sq. in., diameter A_1 and $A_2=$ an area of 82.291 sq. in., diameter B_1 and $B_2=$ an area of 316.647 sq. in., diameter C_1 and $C_2=$ an area of 316.647 sq. in., P= pressure due to delivery head = 203.82 lb., and $P_1=$ pressure due to suction head = 11.18 lb.:

Therefore the out-of-balance load or thrust towards motor =

$$\left[(A_1 - a)P_1 + (C_1 - b) \times \left(\frac{P}{2} - P_1 \right) + (B_2 + A_2)P + (A_2 - a) \right]$$

$$\times \left(\frac{P}{2} - P_1 \right) - \left[(B_1 - A_1) \times \left(\frac{P}{2} - P_1 \right) + (C_2 - b) P \right] = 1,082.847 \text{ lb.}$$

The slightest leak, however, between stages or between neck rings and intake will affect this thrust enormously.

The division plate between the stages is held in place by four small pieces of plate resting in a groove on one side and fastened with four set screws on the other.

A sledge was designed and built to carry the sinking plant, comprising pumps, motors, liquid starters, telephones, instruments and tools. The foundations of the sledge were three runners 40 ft. in length, made up from 8-in. by $3\frac{1}{2}$ -in. channel bars, with 8-in. by $3\frac{1}{8}$ -in. steel plates riveted to the under edge. Both ends of all three runners were curved upwards so that the sledge would clear any reasonable obstruction in the shaft. The two outer runners were built in the form of a warren girder, having four spans of 9 ft., the top

chord of which was made of $3\frac{1}{2}$ -in. by $3\frac{1}{2}$ -in. by $\frac{1}{2}$ -in. angle iron. The design of the sledge lent itself to great flexibility and the bracings of the warren girders, when decked over, formed four suitable platforms to accommodate the sinking plant. Figure 10 shows the general arrangement of the sledge.

On the west side of the shaft, 3 ft. 3 in. was allowed to give room for dismantling the pumps, while on the east side 5 ft. to 6 ft. was left to permit of a skip passing to the water's edge, so that any silt or other material likely to be encountered could be hoisted to the top of the incline. This left approximately 17 ft. for the width of the sledge.

It will be observed that the sledge was designed resting on sills. This was only a tentative proposal, as the probabilities were that the sills had been removed when the Simmer Deep closed down. This proved to be the case, for out of a total distance of 3,052 feet to be traversed on the incline, sills were only met with in 900 feet and these were greatly deteriorated.

Two smaller sledges were placed in front of the main sledge secured to the under-frame by bolts. They were for carrying the suction valves and pipes, and were so arranged as not to foul any dividers that remained in the shaft. The method of attachment of the smaller sledges to the main sledge permitted of their movement in a vertical direction, thus allowing of the foot valve being raised and lowered, but the limit of lowering was 2 ft. 6 in. above foot wall.

The sledge was built on the surface and riveted in sections suitably proportioned for transportation underground. When fully equipped with the pumping plant the total weight was in the neighbourhood of thirty-five tons.

The method of anchoring, lowering or raising the sledge was to wedge three large blue gum timbers, 15 in. or more in diameter, between the hanging and the foot wall of the shaft. Wire rope slings were wound round the two outside anchors, to which were attached two sets of 10-ton Herbert Morris chain blocks, each block having sufficient length of chain to allow 25 feet travel. The chain blocks were fastened to the east and west corners of the sledge by means of shackles, so as to more effectively guide the sledge in a straight course when descending.

A 3-ft. grooved pulley, round which passed a 1½-in. wire rope connected to the centre anchor, was fastened to the bridle of the sledge. This rope, having two coils round the timber, could be let out and clamped according to requirements, and served as a precautionary measure against accident while the chain blocks were being shortened.

Since re-equipment of the shaft had to proceed as the level of the water was lowered, the anchors were transferred every 150 feet and formed, when lined with lagging poles, a suitable penthouse under which the pump men could work

with comparative safety.

The suction hoses were 12 in. in diameter by 12 ft. long, and were manufactured at the S. A. Rubber Manufacturing Co.'s works at Howick on a mandrill prepared by the mine. The suction valves were made of standard flanges and mild steel plate, with leather facing on the gate, as shown on Figure 11.

On the delivery side of the sinking pumps were two 10-in. valves, with branches leading into a common delivery column and, although the sketch shows a rigid connection, it was found advisable to connect up with a 10-in. flexible hose pipe to eliminate vibration and facilitate joint making when the sledge was being lowered. With the use of the 10-in. flexible hose on the delivery column it was possible to disconnect, lower the retaining valve, insert an 18-ft. length of column, lower the sledge and have the whole connected up and the pump running within one and a half hours.

On one occasion the hose broke away from its fastening, with the result that the sledge and electrical gear got thoroughly drenched. The drying out occupied twelve hours. The following was devised to prevent a recurrence of such mishap. Two 10-in. by 18-in. nipples were turned, with a corrugated surface on the outside. The corrugations were 1/16-in. deep and ¾-in. pitch. These nipples were inserted, one at each end of the hose pipe, and the whole slowly revolved in the lathe. Each end was then carefully bound with two layers of 3/16-in. galvanized wire, drawn through two hardwood blocks, securely fastened to the slide rest. The binding was then soldered and formed, when finished, a joint which can be recommended for all work of a similar nature.

Power was transmitted to the motors on the sledge by means of a 1-sq.-in. cable, 1,000 feet in length, loosely coiled on a reel which was fixed in position on 35 level. Large holes were drilled through the centre of the reel, permitting of a free circulation of air, which carried away any heat that might be induced in the cable. The fixed end of the cable was drawn through a hole inside the reel, and coupled to an end junction box which, in turn, was fastened to the reel. The connection between the link panel and cable was made by bolts, which could be disconnected and the cable unwound according to requirements. This formed a simple and expeditious arrangement.

As the sinking pumps are capable of a head of only 500 feet, and the total depth of the shaft is 1,900 feet, they had to be supplemented by pumps of a greater head, which will be referred to hereafter as 'secondary pumps.' With a view to their utilization as a permanent pumping plant at the bottom of the shaft on completion of dewatering, for handling the mine drainage, high-lift pumps were decided upon. The pumps purchased were two eight-stage Allen centrifugal pumping sets, each comprising high and low pressure pumps with interconnecting pipe for the two pump casings, and are driven by a 1,500 h.p. Metropolitan Vickers slip ring motor, running at 1,470 revolutions per minute, placed centrally between the two pumps of four stages each. The pumps have a capacity of 100,000 gallons per hour against a manometric head of 1,930 feet.

The manner in which the secondary pumps were used in dewatering will appear in the course of the paper, as they were brought into operation, but the following will perhaps assist readers in getting a general conception of the scheme.

A temporary sump formed of concrete walls was built in the drive nearest to the maximum head of the sinking pump and in close proximity to this sump a secondary pump was installed, which, for convenience of reference, is named 'secondary pump 1'. The sinking pump was then delivering into the main sump at 35 level, *i.e.*, at the head of the Milner incline, where dewatering was commenced.

When the sinking pump had reached its limiting head of 500 feet, it delivered water into the temporary sump, and the secondary pump was then brought into use raising the water from the temporary sump to the main sump on 35 level.

As the sinking pump approached a vertical depth of 1,000 feet, a second temporary sump was built and the other secondary pump installed (secondary pump 2). Between 1,000 feet and 1,500 feet depth the sinking pump delivered into the second temporary sump, from which the water was raised by secondary pump 2 to the main sump on 35 level. Secondary pump 1 was therefore liberated after secondary pump 2 was put into commission, and was subsequently installed at about 1,500 feet level, where again the sinking pump discharged into a temporary sump, from which the secondary set delivered to the main sump on 35 level. This was the general method of procedure throughout the shaft. Only two pumps were in operation in the shaft at one time — a low-lift sinking pump, following the receding water, and a high-lift secondary pump, which was capable of raising water through the entire length of the shaft.

Seeing that the sinking plant has a capacity of 100,000 gallons per hour, and the Rhodes shaft vertical pumping plant, described earlier in this paper, has only a capacity of 75,000 gallons per hour, dewatering would have been unduly prolonged unless the pumping plant at the Rhodes shaft was considerably increased, and, in any case, a stand-by set was desirable. A second unit was therefore procured and was running within three months from the date of the commencement of dewatering. The second unit is indentical with the existing one, with the exception that the casing and end covers are of bronze. Provision had, of course, to be made for additional power supply by increasing the transformer station at the Rhodes (see A on Figure 12).

Pending the arrival of the sinking pumps and so as to avoid loss of time, dewatering was commenced with a standard 10-in. C.I. low-lift pump, obtained locally. By the time the permanent pumps arrived, 55,000,000 gallons of water had

already been removed.

On the 5th July, 1924, the sledge, fully equipped, was lowered to the water's edge, which was then 49 feet above 36 level. The pump delivery was connected to a 10 in. column,

which discharged behind the emergency wall — previously mentioned — at the suction end of the vertical pumping plant (see H, Figure 12). The wall, with its cast iron doors, was a constant source of anxiety on account of an internal pressure, having been designed for external pressure only. Without this wall, however, dewatering would have been rendered much more difficult. Immediately the level of the water permitted, the doors were reinforced by wooden blocks tightly wedged to convenient supports. When 36 level was reached, concrete walls were built in the drive, forming two sumps, one to be used as a settler and the other for storage.

Thirty-six level was reached on July 28th, after the sledge had been lowered three times. With the exception of slight trouble with the foot valve, the sinking plant worked admirably. The leather hinges of the valve became soft and allowed the flap to fall on one side, thereby rendering it useless when priming. After the leather had been replaced by a bronze hinge the valve functioned satisfactorily.

The drive at 36 level was found to be filled up with mine silt and undissolved lime, but as the sinking pump was delivering 100,000 gallons per hour, as against 75,000 gallons by the pump at the Rhodes shaft, some 25,000 gallons per hour were constantly being returned to the sinking pump suction. The second Rhodes pumping set had not yet been installed. The surplus water was utilized in cleaning out the drives and also ensured that the small sump at this period serving the Rhodes shaft pump would not be depleted of water, which would have been disastrous.

Whilst the sledge was proceeding on its journey, concrete walls were built in the drive at positions marked K and L, Figure 12. A 10-in. valve was built in the wall nearest the shaft for the purpose of drawing off the mud from the settling sump, to be conveyed to worked out areas below.

On the 24th August, the temporary sinking pump had reached its maximum head. Fortunately, by this time, the permanent sinking pumps had arrived and were forthwith installed. The walls in the drive had sufficiently set to permit of the use of the permanent sumps at 35 level. The doors on

the emergency wall were removed and a measuring weir was installed at the top of the settling sump into which the column now discharged.

As the head on the sinking pump was at this time only 120 feet, it was necessary to resort to throttling on the delivery side. This occasioned a great deal of trouble with the valves, mostly in the form of broken spindles and, with a view to improvement, a diaphragm having a central hole 21/4-in. in diameter was inserted between the flanges on the delivery side of the valve. This device, although overcoming the trouble, made an unbearable noise in the shaft, and a diaphragm having ten 1-3/16-in. holes was substituted. Six of the holes were plugged with bronze screws, and as the head on the pumps increased, so were the plugs removed. The second diaphragm proved quite satisfactory, but subsequently two other impellers, with the blades reduced 18 per cent, were installed and the diaphragm removed. This last contrivance, though not ideal, allowed of the use of the valves without damage.

The foot wall of 37 level was reached and passed on the

30th September, 1924.

On the 6th October the new plant at the Rhodes shaft was started up, and from now onwards was capable of dealing with the whole of the water from the sinking pumps plus the mine drainage from the higher levels, which amounted to roughly 350,000 gallons per day.

Level 38 was reached on the 26th October, and although the sinking pump was then working under an unthrottled head of only 377 feet, the time had arrived for commencing the installation of the first secondary pump so as to have it in readiness when the sinking pump should reach its ultimate A sump was built, pump chamber cut, foundation prepared, and the pump installed before the sinking pump reached its ultimate head. As will be seen from the dewatering chart (Figure 13), Secondary pump 1 was started up on the 1st December, 1924.

Level 39 was reached on the 21st November. The question then arose whether it would be more advisable to wash the silt from the various levels into the suction of the sinking pump, from whence it would ultimately be pumped into the settling sump on 35 level, or to leave the silt to dry out slowly and then remove it by coco pans. By the latter method dewatering would have been expedited, but the silt would have taken long to dry, and its removal would have been costly. It was therefore decided to adopt the former alternative and the wisdom of this choice was justified by results. This method of handling mine sludge proved to be quick and economical, and the small amount of damage done to the pumps was surprising. As the silt rose in the settling sump at 35 level it was flushed out into the worked-out areas below.

Only the low-pressure side of secondary pump 1 was installed in 38 level chamber, in four stages, equal to a head of 965 feet. Under these conditions it was necessary either to remove some of the impellers or to run under a throttled valve. The former was deprecated by the makers. The pump was run for two days with throttled valve but, after careful consideration, it was decided to remove two impellers and no trouble whatever was experienced.

About this time the thrust bearings on the sinking pumps began to give trouble, and after they had been re-metalled several times without any improvement it was thought advisable to replace them with ball-bearing thrusts. The ball-bearings were a great improvement over the grooved thrusts, but they also eventually gave out. Examination of the pumps showed very little wear and the problem seemed difficult of solution. It was only possible to take a direct pressure reading on one side of the first stage, which was unfortunate, as otherwise the trouble would have been found out immediately. The design of the pump made it practically impossible to drill into the various spaces, and the suggestion is offered to manufacturers that means of gauging pressures on either side of the impellers might with advantage be provided. It was eventually discovered that there was leakage past the division plate, thereby permitting water to pass from the second to the first stage. To correct this, the four small pieces of plate already alluded to were replaced by a split ring having a groove turned on one edge. In this groove was placed a ring made of 3/16-in. diameter gutta percha which, when the division plate was pressed against it, made a perfectly water-tight joint.

cause of the abnormal end thrust had been successfully solved and no further trouble was experienced. It would appear that the leakage was attributable to unequal expansion, which was probably greater on the shell of the pump than on the division plate. Except in this minor particular the pumps gave no trouble whatever and cannot be too highly commended.

Forty level was passed on the 24th December, 1924, and 41 level on the 14th January, 1925. It was decided to instal the low-lift side of secondary pump 2 on this level. The sinking pumps reached their maximum head on the 5th February, but the secondary pump was not ready to run until the 14th of the month. On that date, the column having been connected right through to the main sump on 35 level, the pump was started up under a throttled head of 950 feet.

Forty-two level was passed on the 2nd March and 43 level on the 3rd April, while the foot wall of 44 level was reached on the 23rd of the month. It was decided to instal a secondary pump in this drive, as it already contained a small pump chamber and sump which had been in use before the Simmer Deep closed down. As the manometric head for the secondary pump had now reached approximately 1,300 feet, it was necessary to instal both low and high pressure sides. The low-lift side of secondary pump 1 was brought down from 38 level (it was no longer required at that point when secondary pump 2 was brought into operation) and along with its high pressure side, which had not as yet been used, was installed in this chamber. Two impellers were removed from the low pressure side and, with the delivery column discharging into the measuring weir on 35 level, the pump was started up on the 15th May.

On the 20th May dewatering had to be stopped for six days in order thoroughly to wash out the drives, and on the 4th June the shaft was found to be blocked with timber, rails, pipes, rocks, mine silt and mud, which had to be removed by means of chain blocks, skips, pick and shovel. The shaft was cleared early in July. This was the most serious obstruction yet met with, though there had been many others of a minor nature.

Forty-five station was passed on the 8th July and, as there was only one stope between this level and the bottom of the shaft, it was considered advisable to instal a secondary pump. A station was cut and a large stope between 44 and 45 levels, with a portion of the drive, converted into a sump having a storage capacity of 3,500,000 gallons. The low-lift side of secondary pump 2 was transferred from 41 level and installed along with the high pressure side of the set (which had not yet been used) in this station. In addition, one of the sinking pumps was removed from the sledge and also installed in the station as an auxiliary, in case of breakdown to the high-lift unit. In such event, the low-lift pump would deliver into the sump on 44 level from whence the water could be raised to the main sump.

Very little trouble was experienced with the secondary units, considering the large quantities of mud and silt which these pumps had to handle under quite appreciable heads. It may interest readers to know that the combination Allen pumps and Metropolitan-Vickers motor were so finely balanced that vibration was imperceptible and, for temporary use, foundation bolts were considered unnecessary. On the last two foundations no foundation bolts were installed.

The sledge reached 1,576 feet below datum on the 30th September and was then brought to a standstill by reason of the shaft being solidly blocked with mine silt. An attempt was made to remove this by means of ordinary sand pumps working in stages, but unfortunately the percentage of sand in the silt was insufficient to keep the large stones in suspension, and they constantly choked the 6-in. delivery column. The scheme had to be abandoned and the tedious method of pick, shovel and bucket resorted to. Dewatering by means of the sledge could no longer be effectively continued and dewatering proper may therefore be said to have ceased on the above date.

The length of the shaft that had to be cleared before the bottom was reached was 325 feet.

There remained some 5,000,000 gallons in the flooded area, which was lodged in a stope adjacent to the blocked portion of the shaft, and precautions had to be taken to prevent this water from breaking in on the workers who were clearing the shaft. Accordingly, an air pump was installed in the

stope, which took out the water as the clearing of the shaft proceeded, so that the risk of flooding was eliminated. The removal of the silt was naturally a slow process and it was not until the latter end of March, 1926, that the bottom of the shaft was reached, some six months after the sledge pumping set was taken out of commission.

Neutralisation of the water was carried out with great care and every possible precaution taken to prevent acid water entering the dewatering system. Lime was dissolved in a large iron tank placed at the head of the incline, and the milk of lime was drawn off by a 2-in. pipe which discharged into the sinking pump suction. In addition, lime was added

MINE: SIMMER AND JACK MINES, LIMITED

Pumping plant at Milner incline (sinking pumps) to 35 level.

Return for period July, 1924, to 28th May, 1925.

DETAILED COSTS

	Running	Maintenance	Neutralisation
White wages	£2,794 10 7 543 12 10 249 11 1	=	
Engineering department expenses Workshops—Carpenters. "—Fitters "—Blacksmiths "—Boilermakers "—Flectricians		£850 3 8	
" —Electricians.) Electric power expenses Stores Lime Assaying Hoisting.	7,063 14 8 108 14 0 	891 6 2	= = =
Totals	£10,983 9 11	£1,741 9 10	
Cost per thousand gallons	5.011d	. 795d.	_

Remarks: Total cost: £12,724 19s. 9d.=5.806d. per thousand gallons.

to the water at convenient places throughout the mine. The acidity of the water ranged from .027 per cent to .150 per cent which, by the above means, was reduced to .003 per cent. The daily consumption of lime was about 10,000 lb. Figure 14 gives the total cost of neutralisation, which, for convenience, is debited against the main pumping plant at the Rhodes shaft, although, as mentioned above, neutralisation was carried out at various places. Figure 14 also shows the total pumping costs during the dewatering period.

MINE: SIMMER AND JACK MINES, LIMITED

Pumping plant at Rhodes shaft, 36th level to surface.

Return for period July, 1924, to 28th May, 1925.

DETAILED COSTS

	Running	Maintenance	Neutralisation
White wages	£1,342 9 5 125 12 11 63 1 10	£ 352 5 10 125 11 7 52 7 0	£130 10 4 608 1 4 320 0 4
expenses			_
" —Fitters " —Blacksmiths " —Boilermakers		1,137 14 11	28 0 11
" —Electricians) Electric power expenses. Stores. Lime Assaying	26,243 16 1 74 9 4 — 13 3 0	2,062 16 5 —	307 4 0 4,221 13 2
Totals	£27,862 12 7	£3,730 15 9	£5,615 10 1
Cost per thousand gallons	10,547d.	1.412d.	2.126d.

Remarks: Total cost £37,208 18s. 5d.=1s. 2.085d. per thousand gallons.

Figure 14b

The following general observations should prove of interest.

On one occasion the 10-in. delivery hose pipe burst and flooded the whole of the electrical gear of the sledge. The burst was not due to any defect in the pipe but solely to the rough usage it had to undergo. So as to avoid a similar occurrence, the hose pipe was reluctantly eliminated and replaced by a 10-in. bronze expansion joint. This was not nearly so convenient for coupling up and after several attempts it also had to be dispensed with. The difficulty was eventually overcome by the installation of a third 10-ton chain block in place of the central anchor rope already referred to.

One of the greatest dangers that had to be contended with was the sudden rising of the water when the sledge was being lowered, which was found to take place invariably when nearing a level. This was discovered to be due to silt in the drives, which dammed back the water until the pressure due to head overcame the resistance of the silt. The silt was then forced out and was followed by a large volume of water that rushed into the shaft. To guard against this, the sledge was stopped two feet from the water's edge, but on one occasion the level rose nine feet, flooded the lower pump and motor, and only stopped rising when two inches under the stator of the second motor.

The ordinary wedge gate valve having cast steel casings and bronze seatings was used throughout. Unfortunately, the seatings on most of these valves gave a great deal of trouble, and approximately 75 per cent had to be reseated in the mine workshops. Some manufacturers screw the seatings into the casings, but the large majority rely entirely on pressing such seats into place and, in one exceptional case, it was found that the seatings were held in a recess ¼-in. in depth. It is impossible, of course, for a valve to operate successfully under these conditions, and it would be an advantage to create a standard having the seatings screwed into the casing, which would give assurance that the valve would function when required.

When the lower levels of the mine were reached, the ordinary cast iron pipe fittings became unsuitable on account of the high pressures and, as it was impossible before actually

arriving at the station in which the secondary pump was to be installed to forecast the fittings which would be required, it was found necessary to have special fittings cast in bronze. These were all supplied by one of the local foundries on very short notice and gave excellent results.

As the ordinary jointing material was unsuitable for high pressure, a \(^5\)%-in. steel plate ring was designed having a groove on each side in which grooves were placed a 5/16-in. diameter gutta percha. When the ring is placed between the fittings to be connected it forms a very simple, strong and water-tight joint.

Figure 15 shows the sledge, with its pumping equipment, in the shaft.

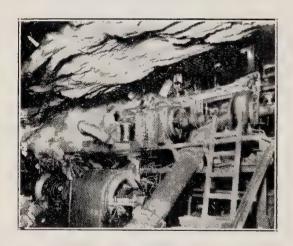


Figure 15

Figure 16 shows one of the secondary pumps installed in 44 level.

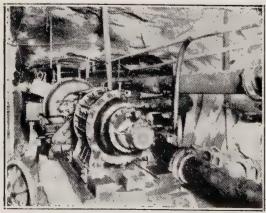


Figure 16

Figures 17 and 18 are two views of the present pumping equipment at the bottom of the Rhodes shaft.

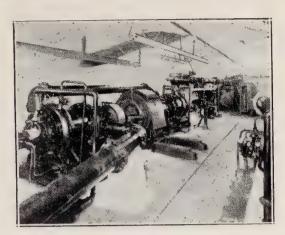


Figure 17

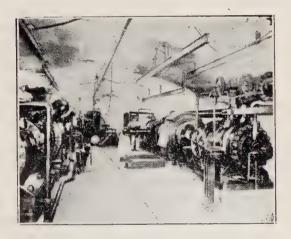


Figure 18

Dewatering, with the exception of 5,000,000 gallons, was accomplished in fifteen months. The sledge with its pumping equipment had traversed some 2,730 feet down the Milner incline, and some 526,000,000 gallons of water had been taken out of the flooded area. It was a time of anxiety to all concerned. Uncertainty loomed ahead, for no one knew what the receding water might reveal. It is very gratifying, however, to be able to record that few accidents occurred. The writer cannot speak too highly of those associated with the work, to whose initiative, resourcefulness and energy its successful achievement was largely due.

DISCUSSION

MR. R. C. CAMPBELL-JOHNSTON (Canada): The dewatering methods described in this paper are of particular interest to us in connection with the problems we have in the alluvial gold mines of Barkerville, Lightning Creek, and other places in British Columbia where it is necessary to control the water. I think the paper gives us the information we require. There should be no difficulty in raising the water in these mines, and so being able to get to the deep-level alluvial deposits in the Cariboo.

PROFESSOR S. J. TRUSCOTT (England): One very interesting point is the great height for a single lift. The author said that the lift was 3,252 feet. I do not know of any greater distance than that. In the iron mines of the Minnesota district, they lift against a head of 2,400 feet, but there is not within my knowledge so high a lift as 3,252 feet. It was reached with centrifugal pumps and it is interesting to note that only 15 impellers were required, giving to each impeller a height of 250 feet. That indicates the improvement of technique in pump manufacture, because a pump that will do 150 feet per impeller is really good.

There is another interesting point. This mine consists of two portions, an outcrop portion and a deep-level portion. These formerly belonged to two different companies, and, the lower portion being abandoned, the outcrop company took advantage of the position by letting its water pour into the abandoned workings where there was sufficient sump to allow them to do without pumping for a period of two years. At a later date the outcrop company was able to purchase the deep-level property, but then they had to take out the water which formerly they had allowed to pour in.

THE TIPPING AND GUIDING OF VERTICAL SKIPS*

By G. W. Sharp (Member, S. Af. Inst. Eng.) **

(Quebec, Que., Meeting, September 26th, 1927)

The wear and tear and general condition of rope, guides, shaft timbering, winding engine, and all that is accessory to the working of the skip, as well as the skip itself, depend so much upon the manner in which it is guided and tipped that practical improvement in this connection must be reflected in no small reduction of working costs. The object of this paper is to indicate the cause of excessive wear and tear and to show by what means improvement can be effected.

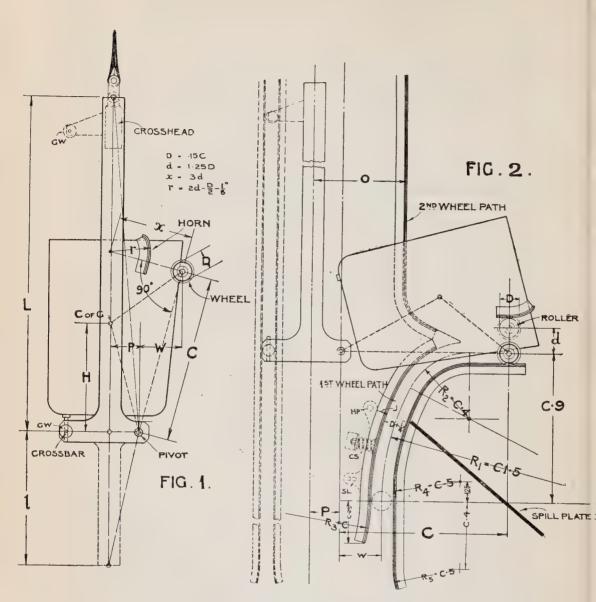
The skip and tipping arrangement dealt with here is the type in general use on the Rand as outlined in Figures

1 and 2.

Other details of construction being immaterial to our investigations, Figure 1 shows merely the arrangement of parts necessary to effect guiding and tipping. The figure represents two drawbars connected at the top by a main crosshead to which is attached the hoisting rope and its connections, and at the bottom by the tipping axle and lower crossbar. The skip body is supported on the two latter, but is attached only to the tipping axle, on which it is free to turn, being fitted with bearings provided with suitable means of lubrication to avoid undue friction. This feature will be referred to as the 'pivot'. The upper part of the skip body on the tipping side is fitted with a pair of wheels and horns or skid bars. The centre of gravity of load and skip body with its attached parts, exclusive of drawbar, is indicated in its ideal position on the centre line of the latter. condition, being so near the truth in a properly designed skip, will be assumed as a fact in these investigations. Letters H, C, P, L, etc., representing dimensions in the arrangement of details will be referred to later.

**Chief Draughtsman, New Consolidated Goldfields, Limited, Johannesburg, South Africa. (521)

^{*}This paper was published in Journal of the S. At. Inst. Eng., XXI,



Skip and Tipping Arrangement.

Special attention is directed to the triangle shown in dotted lines connecting the pivot, centre of gravity, and wheel. This will be referred to as the 'skip triangle', and represents an imaginary frame supporting the internal strains resulting from the process of tipping.

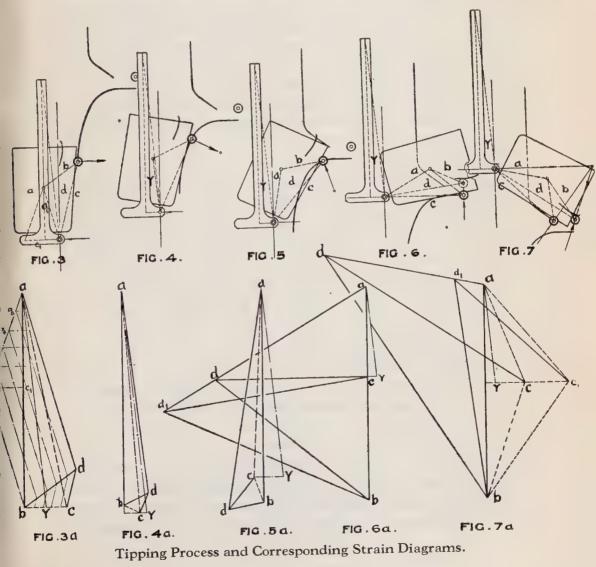
Figure 2 is the type of tipping arrangement in common use on these fields, being known as the Kimberley tip. It consists of two pairs of wheel paths, with which the skip

wheels engage; the paths providing the means of turning the skip body on its pivot. For purposes of reference, these are called the first and second wheel paths. In order to guide the skip wheels from one path to another, rollers are arranged with which the skip horns or skidbars engage in negotiating this portion of the tip.

In enquiry into the tipping process, the writer has made use of graphic statics to determine the reactions, etc., that

take place.

In Figures 3 to 7, the skip and skip triangle are shown in five positions during the tipping process, the wheel paths being indicated by plain lines representing the path taken



by the wheel centres. Below each figure is shown the corresponding strain diagram for that particular position of the skip—Figures 3a to 7a.

The skip triangle is lettered according to the system known as Bow's notation, *ab* representing the load at the centre of gravity, *bc* the reaction at the wheel, and *ca* that at the pivot; while the panel space is lettered *d*, giving to each side of the triangle its notation *ad*, *bd*, and *cd*. The frame diagram of the drawbar is represented by a further triangle and is lettered Y, shown more clearly in Figure 7.

In the strain diagram, the vertical line ab represents to scale the load, and the full lines ad, bd, and cd, the strains in those members of the skip triangle, being, of course, parallel to the direction of each at the instant in question. bc drawn parallel to its known direction will represent the reaction at the wheel, and ca the resulting reaction at the pivot. These reactions are indicated in dotted lines. If we introduce aY into the diagram parallel to the imaginary sloping member of the drawbar, this will represent the resultant of component strains in the drawbar.

The reaction at the top of the drawbar will, under all conditions, consist of two components, viz., that acting horizontally from the guide and that acting vertically from the hoisting rope; similarly, at the bottom of the drawbar the reaction will be constantly horizontal. For our purpose it will be convenient to assume that this bottom reaction occurs horizontally in line with the pivot.

From these strain diagrams it will be seen that:

The vertical component of

aY = the pull on the rope.

The horizontal component of

aY = the pressure on guide at top of drawbar.

cY = the pressure on guide at bottom of drawbar.

ac = the load on the pivot.

bc =the load on the wheel.

Referring to Figures 3 and 3a, the latter shows that, at the moment of entering the tip, sudden pressures, which immediately before did not exist, are put upon the guide at top and bottom of drawbar. These pressures are represented

by aY and cY respectively, while the sudden load on the wheel is represented by bc. In this connection a further triangle with additional lettering a_1 and c_1 is shown in Figure This represents an imaginary frame supporting the load before entering the tip. The faint lines in the strain diagram aa_1 , c_1a_1 , and ba_1 show the strains in this frame; also c_1a represents the reactions of bottom crossbar and c_1b the reaction at pivot. Immediately the skip enters the tip, the strains aa_1 and ba_1 disappear, also the reaction c_1a at crossbar; and the pivot load increases instantly from c_1b to ca. other words, the bending stresses in the drawbar arm on the pivot side have been suddenly almost doubled. sudden increase in strain in the lower end of the drawbar is probably largely responsible for the cracks that sometimes occur just above the tee-head in that member. The sudden application of load to the wheel is also a most objectionable feature and accounts for much of the wear that takes place.

Without the introduction of a spring wheel-path or its equivalent at entry to the tip, this sudden application of loads is unavoidable. But by the use of some such device, whereby the reaction bc on the wheel would equal the resistance of the spring, an effect would be produced like that indicated in the strain diagram 3a by the letters a_2 , a_3 , etc., where the faint lines converging at these points show gradual reductions in the members aa_1 and c_1a_1 , and gradual increases in pivot and wheel reactions.

Continuing to follow the tipping process, it will be seen that, neglecting friction, the normal pull on the rope at entry remains for the moment unaltered. It increases only gradually as tipping occurs and, after reaching a maximum at about the position in Figure 4, gradually decreases until, with the full skip, a minimum is reached at the position in Figure 6, where pivot and wheel are in the same horizontal line.

In each of Figures 6 and 7, two skip triangles are shown; resulting from a change in the point of reaction from wheel to roller and roller to wheel respectively; in Figure 6 the horn is engaging the roller and leaving it in Figure 7. The resultant strains are indicated by lines converging at d_1 in the corresponding strain diagrams. Special attention is called to the fact that in each of these positions no change in rope

pull has occurred. Also that in Figure 6a there is no change whatever in reaction resulting from engagement of horn and roller. In Figure 7a, lines Yc_1 and ac_1 indicate increases of guide pressure and pivot load.

To make a thorough investigation giving conclusive evidence of all shocks and excessive work entailed in tipping, it is necessary to examine the process continuously from start to finish. This has involved the plotting of some 15 to 23 strain diagrams in each of the cases dealt with by the writer.

An example of the resulting compound diagram, which I call the 'diagram of reactions', may be of interest, and is illustrated in Figure 8. This is constructed by plotting a

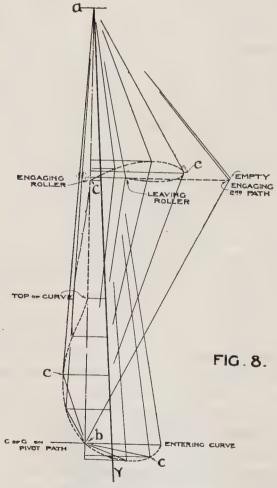


Diagram of Reactions.

series of strain diagrams for the full tipping operation on one load line ab and erasing all lines except those representing reactions.

ab represents the load, and the various lines ac and bc the reactions on pivot and wheel respectively. Points c result in a curve called the 'c curve', shown dotted, which has the virtue of showing errors in graphing, while at the same time indicating in characteristic fashion the changes that take place. After drawing aY representing the imaginary drawbar member, the introduction of the wheel reaction bc at any point will give the measures of all the factors required.

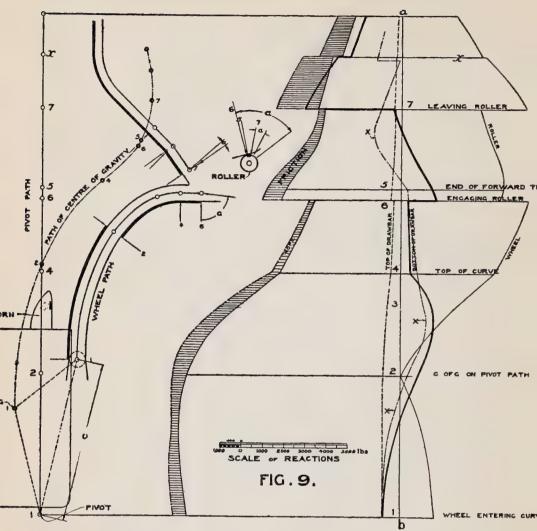
The tipping process commences at the bottom right hand corner on 'wheel entering curve'. At the position centre of gravity or 'c of g on pivot path', only rope pull and guide pressure remain. Where the 'c curve' crosses aY is the point at which pressure on bottom of guide disappears; it will also be noticed that the horizontal component of aY, the pressure at top of guide, is present in varying strength throughout the trip. At the point 'top of curve' a kink occurs in the 'c curve' resulting from the introduction of a portion of horizontal straight wheel path before engaging the roller; a condition easily modified or removed entirely, if desired, by continuing with a suitable curved wheel path until the roller is reached. At the position 'engaging roller', a further distinct change in shape of the 'c curve' results, due to varying directions of wheel reactions taking place without any great variations in rope pull. Between the points 'leaving roller' and 'engaging second path', these representing simultaneous conditions, a horizontal straight line results from a sudden change in direction of reaction bc, representing suddenly increased guide and pivot reactions.

The above method of research will now be applied in the examination of actual cases which have come under the

notice of the writer and which are not exceptional.

Figure 9 is the first case investigated, and will be explained in detail because the method of representation is common to all.

On the left hand is represented the tipping arrangement, the essential features being the first and second wheel paths and the roller. At the bottom, in its position of entering the



Example 1-Tipping and Work Diagrams.

tip, is shown a portion of the skip body, with an outline of the horn, the wheel (shown dotted), and the skip triangle. A vertical line through the pivot indicates the pivot path, while the path of the centre of gravity (c of g) is also shown.

In the passage of the skip through the tipping arrangement there are certain critical positions, these being numbered as follows:

- 1. Entering curve.
- c of g on pivot path.
- Top of curve. 4.
- End of forward travel. 5.
- 6. Engaging roller.
- 7. Leaving roller.

The positions are marked on the three paths by small circles, those on the 'c of g path' being further distinguished by double circles. Other positions showing exceptional features will be also indicated by circles, to which attention will be drawn as occasion arises. The extra heavy lines indicate the surfaces of the several parts which have contact during tipping.

On the right hand of the figures is shown a 'diagram of work'. The vertical line ab is the datum or base line. On either side of the line ab are curves representing, in their horizontal distance from this datum, the magnitude of the reactions occurring during the tipping operation. These curves have been plotted from diagrams of reaction similar to Figure 8 previously referred to. The positions numbered, as well as all intermediate points, are horizontally opposite the corresponding positions of the pivot in the tipping arrangement.

The outside right hand curve (marked 'wheel') represents reaction on the wheel, changing to reaction on roller (marked 'roller') during the period when the latter is working, and again to wheel reaction of the second wheel path. The next curve in heavy full line (marked 'bottom of drawbar') gives the reaction of guide at the bottom of that member, while the curve in heavy dotted line on the left hand side of datum represents the corresponding reactions of guide at top of drawbar and is marked accordingly. These two guide reaction curves are indicated on the side of the datum ab corresponding to the side of the guide from which reactions take place.

The curve, dotted, marked X, will be referred to later. The next curve on the left hand gives the pull on the hoisting rope and is marked 'rope', while the last curve represents increased pull on rope due to guide friction, the latter being indicated by the shaded area marked 'friction.' The area between the extreme left hand curve (rope pull plus friction) and the datum ab gives to scale in ft. lb. the work done during the full operation.

Any friction of wheel path, wheels, or rollers, is regarded as negligible and is accordingly omitted from the writer's calculations. The guides are assumed to be of wood, and the coefficient of friction is assumed to be 0.4, so that the friction indicated in the diagram represents 40 per cent of the sum of top and bottom guide reactions at any position in the journey.

As regards the alteration of load during the period of rock discharge, from observation made by the writer it was found that discharge generally commences when the skip body has reached the end of its forward travel (position 5 in diagram), and is empty when the normal centre line of the body is at an angle of 40 degrees with the horizontal. This condition of discharge is assumed as a fact in The rate of discharge is also a necessary investigations. factor in calculation and, in order to be able to make an adjustment in these curves agreeing as closely as possible with actual fact, the writer has assumed as reasonable that between the start and finish of discharge the rate is constant. Assuming, for example, that during this period the pivot has travelled ten feet, each full load curve of reaction is reduced in the proportion of one-tenth of the rock load at every foot traversed by the pivot between positions No. 5 and the empty position. In order that approximately truthful comparison may be made, and no case accorded more favourable treatment than another, the work diagram in each case dealt with has been adjusted strictly in accordance with the above rule, after, of course, ascertaining the period of discharge.

The curve of pivot reaction will have characteristics similar to the rope reaction curve, and, as its introduction into the diagram might have resulted in confusion with the latter, it has been purposely omitted.

As regards the scales used in these diagrams, in the skip and tipping arrangement the distance C between pivot and wheel centres is five feet, and all other dimensions are in correct proportion.

The length of drawbar or guide slipper is assumed to be $= C \times 2$.

The scale used for the reaction curves is shown on the diagram, its overall length representing 6,000 lb.

The loads in all cases illustrated have been assumed as 7,000 lb. of rock plus 3,000 lb. for skip body, making a total weight acting at centre of gravity of 10,000 lb. The weight of drawbar, shackle, etc., is omitted from calculations.

In Figure 9, the writer would draw special attention to the arrangement of horn and wheel. It will be seen that throughout the forward travel the horn on this skip will always be considerably in advance of the wheel, a relation which will be found to make tipping without shock well-nigh impossible. This arrangement is in effect similar to that in which the top portion of front body or lip plate above the wheels is made to serve the purpose of a horn.

It might be well here to consider the method usually adopted in the design of a tipping arrangement, which is somewhat as follows: Some judgment is used, generally based on conventional practice, regarding the type of horn. Certain arbitrary proportions are then fixed for size of wheel and roller. Some shape of wheel path is adopted which is committed to paper, and by moving a diagram of the skip body along this appointed path on the drawing, the locus of the centre of gravity is plotted. Arriving at the roller, some little difficulty is encountered in an endeavour to maintain regularity in the path of the centre of gravity, which is inclined to show kinks and bumps, and, after a few trials the roller is located in a position giving a more or less regular curve. Then, plotting the locus of the wheel while the roller is in action, the second wheel path is arranged and continued until the skip is in the position of being empty. Having made sure that in this last position the skip mouth is sufficiently covering the spill plate or chute underneath to avoid spillage into the shaft, the general lines of the design are complete, with a result more or less of the nature of that illustrated in Figure 9.

Referring to the curve of wheel reactions in the work diagram of Figure 9, it is seen that, after receiving its initial load at entry, the wheel is gradually relieved until at position 2 it is entirely without load. Reaction then starts from an opposite direction until it becomes a maximum at position 6.

Here it is relieved by roller and horn, when it will be noticed that the reaction suddenly decreases, a sudden rise occurring later as the second wheel path comes into play. The general characteristics of this curve will be similar in all cases.

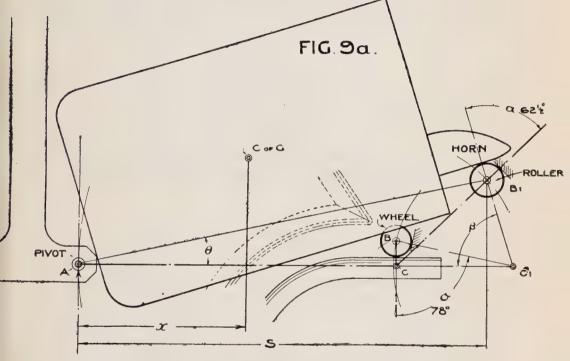
Perhaps the most irregular curve in the diagram is that of horizontal reaction at 'bottom of drawbar'. Starting on one side of the guide with 800 lb., it gradually falls, and changing at zero gently to the opposite side rises to a maximum of 1.530 lb. at a position between 2 and 4. At 'top of curve' it continues with comparatively slow reduction until, on 'engaging roller' it suddenly rises from 320 lb. to 1,670 lb., representing direct shock on the guides of 1,350 lb. passing quietly over to the other side and rising at position 7 to 380 lb., a sudden rise to 4,540 lb. takes place, resulting in direct shock to the guides of 4,160 lb. Then, a little further on, due to sudden change in inclination of second wheel path, position X, sudden reduction of from 3,000 lb. to 2,400 lb. takes place. It will be noticed that between these two last sudden changes an average load on the guides of about 3,570 lb. has been sustained.

Let us now examine the conditions affecting the hoisting rope, undoubtedly the most important item of all. Having assumed that the wheel has entered its path under the best conditions, that is, without any bump, the diagram shows no sudden rise in pull on the rope. It commences the tipping operation with its normal working load of 10,000 lb. (drawbar, etc., neglected). Gradual increase of a few hundred pounds occurs, and, after reaching the maximum, the load is gradually reduced until the pivot is in position No. 6. Here, in the writer's opinion, the case becomes rather serious. From a load of 4,010 lb. there is a sudden rise to 5,590 lb. representing direct impact on the rope of 1,580 lb. If to this be added the sudden increase in friction of 590 lb., the shock becomes 2,170 lb. At position No. 7 a shock of 300 lb. occurs, which becomes 1,960 lb. with the addition of suddenly increased friction. There have been, including friction, two shocks of roughly one ton each in quick succession.

It is interesting to note the sudden fall in rope load occurring at position X corresponding to sudden change in direction of wheel path. This fall measures 410 lb., which,

on the return of the entirely empty skip, will be reduced, resulting in a shock of about 220 lb. to the rope on the start of the downward journey.

It will be fitting at this stage to examine the causes of the shocks above indicated. Attention is directed to the angle α , which is called the 'angle of shock'. This represents the difference between the directions of reactions as they should be, and as they actually are.



Shock in Engagement with Roller.

Figure 9a shows, to an enlarged scale, the skip in the position of engaging roller. Supposing the skip to be moving upward, the motion of the pivot at A will be vertical, and that of the wheel at B in this particular case will be horizontal. Perpendicular to these directions of motion lines are drawn intersecting at C. It will be clear that at the instant in question the skip body and all attached parts are rotating on C as their centre. Drawing B_1C through the centre of reaction of the roller, we have described a perpendicular to the direction of motion of any point on B_1C . We now draw B_1C_1 in

the direction of reaction resulting from contact of horn and roller. The angle CB_1C_1 , or angle α , will be the 'angle of shock'. The effect of this angle is somewhat similar to that which would occur on a level railway if sudden upward inclination of track were introduced. In that case the angle α would be the angle of the sudden incline resulting in direct impact to traffic.

In the diagram, the velocities at each point of reaction A, B, and B₁ will vary directly as the lengths of the radius vectors AC, BC, and B₁C respectively; the forces of impact at A and B₁ will vary inversely as their velocities; and their moments about the centre or axis of rotation C will be equal.

Therefore, the force of impact on the rope or pivot at A is obviously the direct result of resistance to motion causing impact at B_1 .

This resistance to motion at B_1 is proportionate to that component of reaction acting in the direction of motion; or is equal to the reaction at impact multiplied by $\sin \alpha$.

To arrive at the value of these shocks by formulae, we have:

- S = Horizontal span between the pivot and the point of reaction at which shock occurs, in feet.
- x = Horizontal distance between pivot and C of G (centre of gravity), in feet.
- α = Angle of shock.
- β = Angle of reaction at roller or wheel, wherever shock occurs.
- θ = Angle formed by AB₁ and the horizontal.

Then, where:

w =Weight acting at C of G, in lb.

R_w = Reaction in lb. of wheel or roller, whichever is receiving shock.

 R_r = Pull on hoisting rope after impact, in lb.

 F_r = Force of impact on hoisting rope, in lb.

 $F_{\rm w}$ = Force of impact on wheel or roller, in lb.

We get:

$$R_{w} = \frac{wx \tan \beta}{S (\tan \beta + \tan \theta) \sin \beta}$$

$$R_{r} = w - R_{w} \sin \beta = w - \frac{wx \tan \beta}{S (\tan \beta + \tan \theta)}$$

$$F_{w} = R_{w} \sin \alpha$$

$$F_{r} = \frac{R_{w} \sin \alpha \sin \theta}{\sin (\alpha + \beta + \theta)}$$

In Figure 9c (Cases I., II., III., etc.), are shown six variations in positions of skip and directions of wheel or roller reaction; and the above formulae, modified to suit, appear below each variation for upward or downward journey.

Defining impact as the product of force F acting through a distance H, we have:

$$FH = \frac{Fv^2}{2g} \text{ in ft. lb.}$$

Applying this to any case being dealt with, let V_r and V_w =velocities at pivot and wheel or horn, respectively, when shock occurs,

$$\frac{F_r}{V} = \frac{F_w}{V_r}$$

The impact on hoisting rope, neglecting guide friction:

$$I_{\mathrm{r}}\!=\frac{F_{\mathrm{r}}^{\phantom{\mathrm{r}}}V_{\mathrm{r}}^{\phantom{\mathrm{r}}^{2}}}{2g}$$

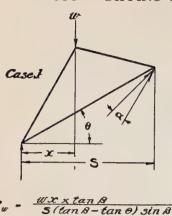
Impact on wheel or roller, at whichever of these shock occurs:

$$I_w = \frac{F_w \ V_w^{\ 2}}{2g} = \frac{(F_r \times V_r)^2}{F_w \times 2g}$$

The above formulae, derived from the graphic method, afford ready means to finding shock, etc., in any existing tipping arrangement. They do not, however, give the shock on rope guides and the resulting increase to impact on rope from added friction, which depend on the proportions of drawbar, and also on the conditions immediately preceding shock. The latter are easily found by graphic methods; but, when introduced into formulae for finding guide shock, make them too complicated to be of practical use.

So as to make comparison fair, while at the same time simplifying calculation in the examples dealt with here, the writer has assumed V, the velocity of the pivot or drawbar at the end of forward travel, to be 8 ft. per second, and retardation and acceleration 3 ft. per sec. per sec. This assumption will, of course, not be true for all cases, but is considered for purposes of comparison to be a reasonable average.

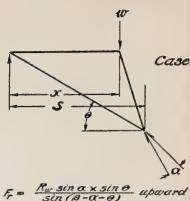
TIPPING AND GUIDING VERTICAL SKIPS—SHARP 536



upward & downward.

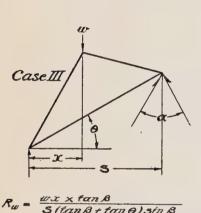
Fw = Rwsina

upward only



$$F = \frac{R_w \sin \alpha \times \sin \theta}{\sin (\beta - \alpha - \theta)} \text{ upward}$$

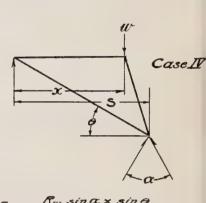
$$F = \frac{R_w \sin \alpha \times \sin \theta}{\sin (\alpha + \beta - \theta)} \text{ downward}$$



W- Ro Sin B = W- Wx x tan B

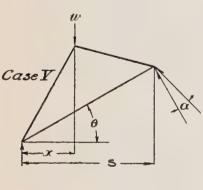
$$R_{w} = \frac{w \times \times \tan \beta}{S(\tan \beta + \tan \theta) \sin \beta}$$

$$R_{r} = w - R_{w} \sin \beta = w - \frac{w \times \times \tan \beta}{S(\tan \beta + \tan \theta)}$$



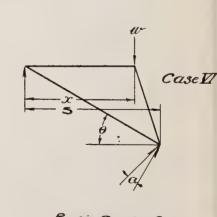
 $\frac{R_w \sin \alpha \times \sin \theta}{\sin (\alpha + \beta + \theta)} \text{ upwara}$ $\frac{R_{\omega} \sin \alpha \times \sin \theta}{\sin (\alpha + \beta - \theta)} downwan$

For downward journey change the signs in the denominator.



wx x tan B

Fw - Rw sin a



 $\frac{R_w \sin \alpha \times \sin \theta}{\sin (\alpha + \beta + \theta)} \quad upware$ S(tan 8+tan 0) sin B upward & R, = W-Rw SINB = W- Wx x tan B downward Rw sin a x sin 0 downwar S(tanB+tan0)

FIG. 9c.

Shock Formulae.

In the statements given later, referring to the examples illustrated herein, the distance H through which the force of impact acts is expressed in inches.

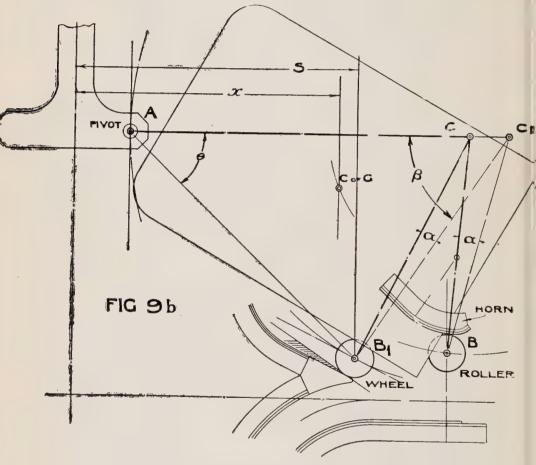
Referring again to Figure 9a, it will be clear that on the downward journey the centre or axis of rotation becomes C_1 , resulting from the direction of motion of horn on roller; the direction of reaction being B_1C_1 . Consequently, BC being the direction of reaction at wheel, the angle of shock a is the angle CBC_1 , and the wheel is the point at which shock occurs. The hoisting rope at the instant in question will experience a sudden reduction of strain.

The shaded lines at roller and wheel represent the directions in which either the horn or path, respectively, should make contact to avoid shock. It is obvious that, had either path or horn been constructed to give the direction indicated by these shaded lines, the resulting thrust against the guides would have been enormous. But in avoiding the heavy guide pressure, very considerable shock is imposed on almost everything concerned. It will be shown later that excessive guide pressure, as well as impact from such causes, can be entirely avoided.

In Figure 9b is shown the diagram of a skip in the position of leaving the roller and engaging the second wheel path. The notation adopted in this diagram is the same as that in Figure 9a, so that a description becomes unnecessary. It will be noticed that the construction of skid bar or horn differs considerably from that in Figure 9a. But the particular feature to which the writer would draw attention is the absence of actual shock angles. The shaded line, together with the radius vectors B₁C₁ and BC₁, indicate a direction of path which would, if adopted, result in shock. The practice usually followed is to place the wheel path at this stage more or less on the shaded line, and, whether the practice is deliberate or not, the writer can see no good reason for it. The method of plotting the locus of the wheel up to this point is probably responsible for the introduction of this shock angle, as it is extremely difficult to discover merely by judgment

the correct tangent of the resulting curve. It will be seen that, although the angle α is comparatively small, the ratio $\frac{x}{S}$ may cause the shock to be considerable.

Before passing on to the next case, a brief resumé of the conditions shown in Figure 9 will be of advantage. They are as follows:



Elimination of Shock in Leaving Roller.

Example 1. (Figure 9)

UPWARD JOURNEY

Pos. 6. Angle $\alpha = 65$ degrees.

Impact on rope (neglecting friction) 1,580 lb \times 12 in. = 1,580 ft. lb. " " (including friction) 2,170 lb. \times 12 in. = 2,170 ft. lb. " rollers 4,150 lb. \times 1.74 in. = 600 ft. lb.

Pos. 7. Angle $\alpha = 6\frac{1}{2}$ degrees.

Impact on rope (neglecting friction) 300 lb. × 8.85 in. = 220 ft. lb. " (including friction) 1,960 lb. \times 8.85 in. = 1,440 ft. lb. " wheels 600 lb. \times 1.82 in. = 910 ft. lb.

DOWNWARD JOURNEY

Pos. X. Impact on rope (neglecting friction) 220 lb. × 3.73 in. = 68 ft. lb.

Pos. 7. Angle $\alpha = 19\frac{1}{2}$ degrees.

Impact on rollers 460 lb. \times .6 in. = 23 ft. lb.

Pos. 6. Angle $\alpha = 84$ degrees.

Impact on wheels 1,790 lb. \times .84 in. = 125 ft. lb.

GUIDE REACTIONS AND SHOCKS

At entry 800 lb.; shock.

1.530 lb.; maximum reaction. Pos. 2 to 4.

Pos. 6. 1.350 lb.; shock.

1.670 lb.: maximum reaction.

4.500 lb.: maximum reaction. Pos. 7.

4.160 lb.: shock.

300 lb.; shock (downward journey). Pos. X.

In the example illustrated in Figure 10, the outstanding feature of the design is the shape of the horn. Apart from the results of investigation for shocks, etc., one of the chief objections to this type is the uncertainty of its being constructed accurately to design. The shape employed in this investigation was reduced from the actual template used in manufacture, and it may be reasonably assumed that any inaccuracies in construction, as also the wear and distortion which takes place with use, will increase rather than diminish the shocks and excessive guide pressures appearing in the diagram of work.

Special attention is called to the sudden reversal of guide pressure at position 6, representing a shock on the guides of 3,600 lb. At this point, also, there occurs considerable shock on rope, roller, horn, etc. Again at position X, between 6 and 7, due largely to shape of horn, a guide pressure of 4,870 lb. has been attained resulting from a wide angle in direction of reaction. The angle of shock at position 7 is fortunately so slight as to make rope shock, except that resulting from guide friction, a negligible quantity.

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The following is a statement of shock and other outstanding features:

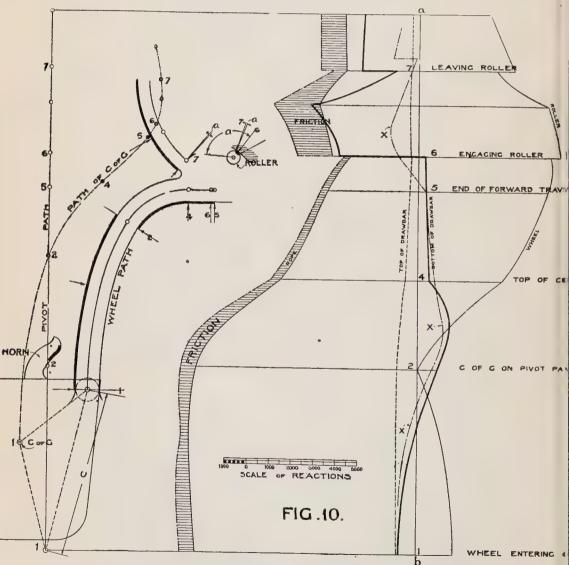
Example 2. (Figure 10)

UPWARD JOURNEY

Pos. 6. Angle $\alpha = 59$ degrees.

Impact on rope (neglecting friction) 820 lb. \times 12 in. = 820 ft. lb. " " (including friction) 2,160 lb. \times 12 in. = 2,160 ft. lb. " rollers 5,650 lb. \times .25 in. = 120 ft. lb.

Pos. 7. Slight impact on rope and wheel.



Example 2—Tipping and Work Diagrams.

DOWNWARD JOURNEY

Pos. 6. Angle a = 34 degrees.

Impact on wheels 1,120 lb. \times 1.1 in. = 100 ft. lb.

GUIDE REACTIONS AND SHOCKS

At entry 800 lb.; shock.

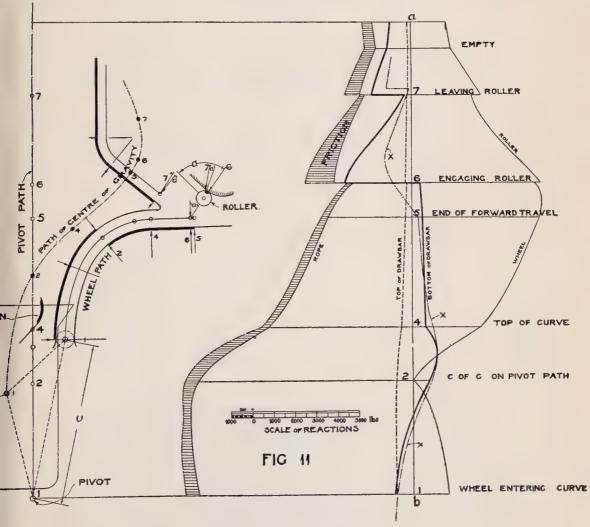
Pos. 2 to 4. 1,430 lb.; maximum reaction.

Pos. 6. 3,600 lb; reverse shock.

Pos. X. 4,870 lb.; maximum reaction.

Pos. 7. 1,580 lb.; shock.

In Figure 11 is shown still another type of horn which, arranged at a different angle combined with a small adjustment of roller, would have resulted in a tipping operation practically



Example 3—Tipping and Work Diagrams.

devoid of shock. Rivetted as it is to the skip body in a position calculated to withstand the strains of reaction, it has the advantage of strength as well as other advantages which will be referred to later (See page 549).

Due to the unfortunate arrangement of horn and roller, rather severe shocks occur at positions 6 and 7. Here, again, in the first of the these, occurs a sudden reversal of guide reaction of considerable magnitude.

The special features of this figure are as follows:

Example 3. (Figure 11)

UPWARD JOURNEY

Pos. 6. Angle $\alpha = 79\frac{1}{2}$ degrees.

Impact on rope (neglecting friction) 880 lb. \times 9.7 in. = 710 ft. lb. " " (including friction) 1,960 lb. \times 9.7 in. = 1,580 ft. lb. " rollers 6,000 lb. \times 0.21 in. = 100 ft. lb.

Pos. 7. Angle $\alpha = 6$ degrees.

Impact on rope (neglecting friction) 250 lb. \times 7.7 in. = 160 ft. lb. " " (including friction) 750 lb. \times 7.7 in. = 480 ft. lb. " wheels 390 lb. \times 3.16 in. = 100 ft. lb.

DOWNWARD JOURNEY

Pos. 7. Impact on rollers 300 lb. \times 1.45 in. = 36 ft. lb. Pos. 6. Impact on wheels 1,150 lb. \times 0.7 in. = 65 ft. lb.

GUIDE REACTIONS AND SHOCKS

At entry 860 lb.; shock.

Pos. 2 to 4. 1,150 lb.; maximum reaction.

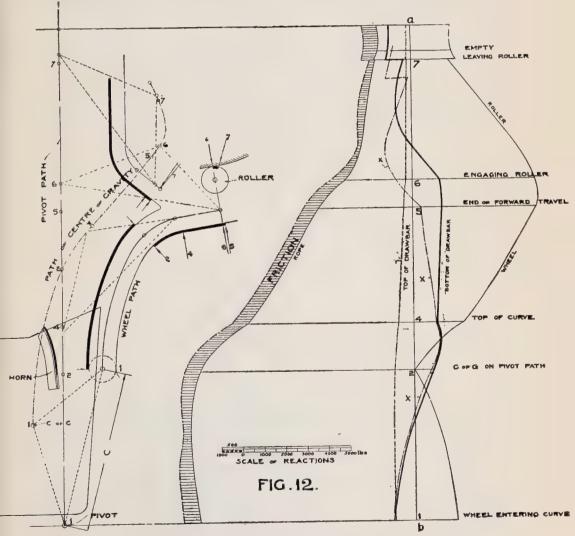
Pos. 6. 3,140 lb.; shock. Pos. 7. 1,550 lb.; shock.

1,880 lb.; maximum reaction.

In Figure 12 is represented a tipping arrangement which shows great improvement on those already considered. It will be observed that, excepting a slight kink in the curve at position 7 'leaving roller', the operation has been performed entirely without shock to the hoisting rope. This is one of six identical tipping arrangements at the Chris shaft of the Robinson Deep, Limited. Two of these arrangements have

only recently been installed, but the other four have been in more or less constant operation during the past 4 years, and have given entire satisfaction to all concerned. As compared with previous tipping arrangements on this mine, the wear and tear and resulting maintenance charges have been considerably reduced. It will be seen that the type of horn, more properly termed skid bar, is similar to that previously shown in Figure 11.

Objection will most likely be raised regarding the shape of the curve of guide reactions at bottom of drawbar. Between the position 4 'top of curve', and 5 'end of forward travel'



Example 4—Tipping and Work Diagrams.

there is a gradual increase in reaction representing a comparatively long-sustained pressure on the guides. This is entirely due to the inclination of the wheel path during this period; an inclination introduced to bring about a reduction of guide pressure during the contact of horn and roller.

Particular attention is called to the relative positions of wheel, horn, and roller. At position 6, where shock in tipping usually occurs, it will be seen that lines drawn perpendicular to the directions of motion of pivot, wheel, and horn, will intersect at centre of roller. The centre of the roller then, in this case, is the centre or axis of rotation of the points referred to. There can be no question that the gliding motion of horn on roller in the upward journey and of wheel on path in the downward journey, are made without shock, a result fully confirmed in practice. In this connection it is interesting to note that this position 6 is passed in both directions of travel without appreciable noise. At position 7 a slight angle of shock is encountered which, as in the preceding cases, could have been avoided.

The following is a summary of outstanding features:

Example 4. (Figure 12)

Pos. 7. Slight Shock to roller and rope.

GUIDE REACTIONS AND SHOCK

At entry 980 lb.; shock.

Pos. 1 to 6. 1,370 lb.; maximum reaction. Pos. 6 to 7. 740 lb.; maximum reaction.

Pos. 7. 820 lb.: shock.

The method used by the writer in the design of the tipping arrangement in Figure 12 was as follows: Instead of relying chiefly on the shape of the path of the centre of gravity as a guide in the arrangement of roller, etc., graphic statics was used to discover and correct indications of excessive work. After a tedious process of trying different shapes of wheel path, as well as several adjustments of horn and roller, the general arrangement shown was adopted.

Elimination of Shock.—By following the proportions given in Figures 1 and 2, which offer a simple rule in design, a tipping arrangement can be directly laid down which, under any conditions of load, shape of body, etc., will give results as nearly as possible approaching an ideal tip.

The diagram of such an arrangement, with corresponding diagram of work, is shown in Figure 13. The diagram of work being self-explanatory, let us consider the design.

As regards the wheel path, the first thing in design is shape and proportion, and in this connection attention is drawn to the curves shown in Figure 13a, which represent the centre lines of possible wheel paths.

A feature common to the first two curves shown is that their vertical axes are at a distance from the pivot path equal to the distance C between centres of pivot and wheel, in order that the position 'top of curve' will coincide with the position 'end of forward travel', resulting in reaction curves devoid of sudden changes. Below each curve is shown the corresponding guide reaction curve at top and bottom of drawbar, these being the reactions chiefly affected during this period of tipping.

The first curve on the left is the quarter of a true ellipse whose major axis is 1.5 times the minor axis; the second or middle curve is the quarter circle; while the last, on the right, is a composite curve of two radii terminating in a straight horizontal line.

An interesting feature of the quarter circle is that the position resulting in maximum horizontal reaction at pivot occurs when the angles γ and β are equal, the latter being the angle of wheel reaction.

Comparing the maximum horizontal reaction of the three curves, it will be observed that the ellipse gives the greatest reaction and the composite the least.

The next point to consider is the relative rate of change in reactions, i.e., whether the work of moving the load or centre of gravity across the pivot path is done too quickly or otherwise. Abrupt action at this point might almost result in shock. In this respect the quarter circle is most unfavourable.

The third curve, which gives least horizontal thrust and has a satisfactory rate of change, is that adopted. It is also very similar to many in use. Expressing its dimensions in terms of the distance C between wheel and pivot centres, they are as shown in the diagram.

Reverting to Figure 13, the next point to consider is the

arrangement of horn and roller to avoid shock.

The portion of horn shown shaded in contact with roller illustrates its position at the instant of leaving the latter.

As regards the engagement of horn and roller on the upward journey, it will, I think, be granted that this is best accomplished when the velocity of the wheel is a minimum. At the end of its forward travel its velocity is nil; this is, therefore, the best position to make the change. A line AC from the pivot perpendicular to its direction of motion will intersect the wheel centre at C, showing this to be the centre of rotation. To entirely avoid shock, the direction of reaction must remain unchanged at any instant; therefore, a line CB is drawn through the centre of rotation in the direction of wheel reaction. The angle ACB will be a right-angle. at any convenient point on CB, the roller is placed, bearing in mind that the less the distance between wheel and roller, the less will be the velocity at which the horn engages. The wheel and roller diameters will here be the controlling feature. The direction of horn surface at point of contact, to maintain unaltered reactions, will be perpendicular to CB, and parallel to AC, the common centre line of wheel and pivot.

As design continues, it is advisable to keep the direction of reaction at roller well restricted, to avoid excessive guide reactions. It is, therefore, necessary to introduce a curve to the horn for this purpose. Except that the wheel has to be lifted as quickly as possible for engagement with the second wheel path, the radius of this curve would be equal to the distance between centres of wheel and roller, less half the diameter of roller. Such a curve would result in least possible reaction. But to lift the wheel sufficiently quickly to get a comparatively small angle of reaction at contact with the second path, the radius of this curve will be:

Twice the distance between wheel and roller centres less half the diameter of roller.

When the wheel has been raised sufficiently to clear the top of the first wheel path, we draw A_1C_1 through the pivot centre perpendicular to its direction of motion, and B_1C_1 through roller centre perpendicular to motion of horn. B_1C_1 is, of course, also the direction of reaction of roller, and C_1 is the centre of rotation. Drawing C_1B_2 through the wheel centre gives the direction at wheel which is perpendicular to the direction of the required second wheel path. In this last stage there is unavoidable shock on guides, although, neglecting friction, there is no impact on the hoisting rope.

Regarding this guide shock at position 7: as before mentioned, to avoid entirely any sudden change in reactions, the direction of reaction must remain unaltered; that is to say, in the diagram, the angle $B_1C_1B_2$ must not exist, a condition impossible to fulfil.

Except for slight shock, due to friction only, and suddenly increased pressure on guides, impact has been entirely eliminated.

The dotted line deviating from the curve of guide reactions between positions 2 and 4 represents the line that would result from the substitution of a straight horizontal track, as in Figure 13a, in place of the finishing arc in the first wheel path. Seeing that this gives no serious irregularity in the guide reactions curve, and would simplify construction, it is adopted by the writer in preference to the curve.

The outline of horn, shown shaded, in contact with roller, indicates its position when leaving the latter.

The dotted lines at the commencement of wheel and guide reaction curves (position No. 1) represent a desirable modification that would result from the use of a spring path at the entry of the tip. It will be seen that, not only would the application of loads on guides and wheels be gradual, but there would also be a reduction of those loads.

In Figures 9, 10, 11, and 12 the curve of guide reactions that would result if horn and roller were arranged according to the writer's rule is shown in each by a dotted line marked X.

Referring again to Figures 1 and 2, as previously mentioned, these furnish a simple rule for laying out any tipping arrangement which will give minimum reactions

and be devoid of shock. Assuming, for example, that the general lines of a skip have been decided upon, and that the distance C between pivot and wheel centres is 5 feet, all that is necessary is to apply the proportions in Figures 1 and 2 as follows:

The position and inclination of second wheel path is easily fixed after finding the centre of rotation of the body at the instant of leaving the roller by the method previously described on page 546.

In view of the conditions on which the proportions of skip body depend, such as the relation between dimensions of shaft compartment and load, it would be impracticable here to attempt to fix ratios for the dimensions of the skip triangle represented by letters in Figure 1. But if the following rules are adhered to, the best results as regards strains and reactions will be obtained.

The ratios $\frac{C}{H}$ and $\frac{W}{H}$ to be as large as possible; $\frac{P}{H}$ to be as small as is safely possible.

It will be seen that the first wheel path is composed of a curve of two radii terminating in a length of straight horizontal track. The skip is shown in the position of engaging roller. The dimension O in Figure 2 will, of course, depend on the shape of the skip body, besides other local conditions. Special attention is called to the flaring of the wheel path at entry. Any inaccuracy either way in the dimension W resulting from wear, causes actual shock at the point of entry, unless this flaring of mouth is of sufficiently large radius.

As regards the wheel diameter, the ratio given in Figure 1 merely represents the writer's idea of proportion, and may be modified to conform to individual opinion without seriously affecting results. In this connection the writer would venture to recommend as of advantage in facilitating repair, as well as construction, the making of wheel and roller to the same pattern.

The type of horn necessary in this design is to be strongly recommended if only for the following reasons: It consists of an angle bar provided with a wearing strip which can be replaced as occasion requires; it is stronger, and provides a much larger bearing surface on the roller, than the older type; in sinking operations, where a projecting horn interferes with the filling of the bucket, this can be arranged out of the way and gives no trouble.

The bottom extension of the side of the first wheel path farthest from the shaft is merely a safety precaution. The engineer occasionally picks up speed a little too quickly at the start, and this extended rail of the wheel path is to counteract any tendency of the skip on such occasions to get away without having righted itself. This extension is sufficiently wide of the wheel to make sure that no fouling occurs at entry on the upward journey.

In dotted lines on the other side of the first wheel path is indicated what would be the writer's method of bringing about a gradual, instead of sudden, application of loads at entry, at the same time further compensating for alteration, through wear, of the dimension W above referred to. J indicates a joint in the path, which latter, from this point downwards, would be hinged at HP (hinge pin) and secured at SL, a lug slotted to allow of the necessary movement. In addition, the rail could be further secured at intervals along its length, the usual rivet holes in the angle bar being slotted, and the rivets replaced by studs or bolts. Between hinge and slotted lug, at a position calculated to give the

required gradual application of loads, would be fixed a strong compression spring CS. The movement necessary would be only small, and the wear at the hinge most likely negligible. With due deference to a prevailing prejudice against springs, the writer is of opinion that the resulting benefit to guides, rope, skip, drawbar, and wheels would more than justify the use of such a device.

The projection on the left at top of drawbar carrying a guide wheel GW, together with a corresponding wheel at bottom on the horizontal centre line of pivot, may be of interest and illustrates an additional feature introduced into the design of the Robinson Deep skips. Just before the skip enters the tip, these wheels engage in a pair of wheel paths, each formed of two angle bars shown dotted in Figure 2. The object here is to relieve the guides entirely during the tipping period. The practical results in reduction of maintenance costs, to say nothing of the saving of time otherwise spent in replacement of worn guides, has thoroughly justified the adoption of this device.

As regards the extension to the drawbar at its lower end, shown by dotted line in Figure 1, its effect on guide reactions is represented by the following simple formulae:

Where:

 G_t =Guide reaction before extension at top of drawbar. G_b =Guide reaction before extension at bottom of drawbar. g_t =Reaction after extension, top of drawbar. g_b =Reaction after extension, bottom of drawbar.

At all times,
$$g_b = \frac{G_b \times L}{L + l}$$

When the reactions occur on opposite sides of guide:

$$g_{t} = G_{t} - \frac{G_{b} \times l}{L + l}$$

When the reactions occur on same side of guide:

$$g_{\rm t} = G_{\rm t} + \frac{G_{\rm b} \times l}{L + l}$$

This shows that an extension reduces the reaction at bottom of drawbar under all circumstances of tipping, but that at top of drawbar a reduction is only effected when the reactions at top and bottom are in opposite directions. When both top and bottom act simultaneously on one side only of the guide, the reduction of pressure or reaction at the bottom is transferred to the top, making that at the latter increase. On reference to the work diagram, it will be seen at what periods of the tipping operation these conditions arise, and the advantage or otherwise of extending the drawbar can be decided to suit the particular case in question.

Effect of Shock, Etc.—As regards the effect of shocks and excessive reactions during the tipping operation, let us consider the hoisting rope as being of first importance. At this stage of winding, the length of rope between skip and sheave is shortest. Near the sheave, elasticity is a negligible factor, and any appreciable impact, though small compared with the breaking strain of the rope, must, if repeated at intervals of a few minutes over long periods, have a very disastrous effect in fatigue in the rope.

The tendency of ropes to fail near the skip more often than elsewhere, and the necessity for frequent re-capping is, in the writer's opinion, very largely due to the shocks in tipping.

Let us suppose we have a 1½-in. diameter rope, with a breaking strain of 60 tons, and capped in approved manner for winding. Having supported it on an 8-ft. diameter sheave, allowing about 15 to 20 feet to hang freely, we make the other end secure, and load the free end with 3 tons. Then suppose that at intervals of five minutes throughout a period of six days the loaded rope be subjected to shocks due to one ton dropping ten inches, and some idea will be had of what happens every week in a very ordinary case of shock in tipping.

An experiment on the above lines would certainly afford most interesting, and at the same time convincing, information. In its absence one can only depend on imagination.

Immediately connected to the rope, and receiving similar shock, will be the safety hook, shackle pins, shackles, and king bolt, representing in effect an ordinary chain. Each pin and link will receive its share of the punishment without any relief of cushioning from an elastic rope.

Even the complete fracture of a heavy shackle pin is not unknown on these fields. The cause of such failure to apparently generously proportioned details must originate in shock, and, excepting the shock in loading which occurs at the end of a long elastic rope, the tipping arrangement is surely responsible.

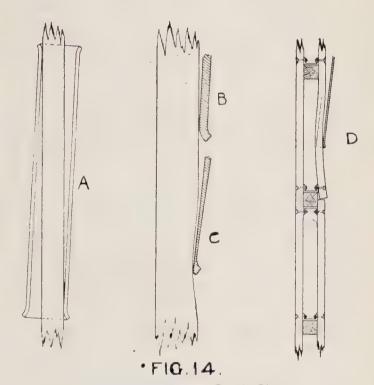
The next link in the chain is the drawbar, and this, though having a cross sectional area far exceeding that required by theory, is often a source of much anxiety. When fracture occurs in this member, it is generally at a point just above the bottom crossbar, a fact responsible for the generous curves shown in Figure 1 at this point. Fracture here is probably due to the twisting strains occasioned by a spinning rope, combined with vibrations resulting from ordinary winding operations, conditions which depend a great deal on the method of guiding in the shaft. But we must look to the tipping arrangement as also largely responsible.

Except at the moment of entering the tip, the severe conditions attending which have been already referred to, the shock chiefly affecting the drawbar will be the same in magnitude and direction as that on the hoisting ropevertically downward—and will create a sudden additional bending stress which must severely tax its power of endurance, especially in cases where the bottom tee head supporting pivot and crossbar is riveted on instead of being forged.

The excessive wear and tear on the guides in the tipping arrangement, as well as on the wheels, horns, rollers, and pivot, are so obviously the result of the tipping operation that particular reference to the desirability for improvement in the conditions affecting them seems unnecessary.

The Guiding of Skips.—Turning now to the question of the guiding of skips in vertical shafts, the problem presents certain adverse factors, namely, the necessary clearance between the fixed guide and the slipper or runner, the spinning of the hoisting rope, the irregularities, slight or otherwise. in the guides, with resulting vibrations.

The common type of long rigid channel guide runner or slipper is provided with wearing strips designed to rub against the fixed guide. Whether these wearing strips are continuous throughout the slipper's entire length, or are fitted over only portions of the length at each end, they will seldom bear against the guide over the full area of their rubbing faces. On the contrary, only a very small area, in extent not more than perhaps three or four square inches, will be bearing at any moment during the trip. This condition, which results from the clearance space between slipper and guide, and the unavoidable eccentricity of the load, is responsible for most of the excessive wear and trouble that occurs.



Action of the Rigid Guide Slipper.

To illustrate this point the writer would refer to Figure 14, which shows, in exaggerated form, the relation of the slipper and guide during the journey up or down the shaft. On the left of the figure is represented, at A, a portion of fixed guide with the channel slipper in its usual position of working. Only the extreme point on opposite sides at either end is bearing. At B and C are shown enlarged views of the lower end of the slipper, B being a new wearing strip,

and C the same strip after being worn. The long taper or bevel resulting from wear is not flat as might be supposed, but, due to increases of clearance and angle resulting from wear, the worn face is slightly curved, somewhat like a chisel might be after being sharpened by an amateur. It requires but a very short period to bring about this result in the guide slipper. There being only a small area in contact, the wear is consequently more rapid than would otherwise be the case.

At portions of the trip where soft places in the wooden guide are encountered, as at C, this rigid chisel-like end of the slipper gradually wears a hollow, the rate of wear accelerating as the hollow is deepened—and the wave of vibration which is started is soon joined by others resulting from similar causes.

A slightly projecting joint brings about a decided bump in the journey, and, a joint or perhaps a disturbed splinter in the guide projecting still further, ripping takes place with all its evils.

Vibration also probably results from the alternate bending and straightening of the guide under concentrated load, and must be very largely responsible for movement at the joints of shaft guides, which the detail at D, Figure 14, attempts to illustrate. It is largely from this cause, in the writer's opinion, that the loosening of bolts and their chewing into the timbering occurs.

In addition to these evils there is the adverse effect of the spinning of the rope.

The tapered wear of the slipper has also a very bad effect in tipping. During those periods where the pressure at top and bottom of drawbar is on the same side of the guide, the slipper, to give the best results, should be able to distribute the load. Instead of distribution, its curved shape results in the load being concentrated.

In any contrivance to overcome the above adverse conditions, there must be flexibility; and a swivel guide slipper having this property combined with strength and durability has been devised and patented by the writer.

Figure 16 shows a skip equipped with this device, also details of the swivel joint.

S is the slipper, provided in the usual way with renewable wearing strips on its three inside working faces. To the back of the slipper is riveted a circular plate trunnion T, the flange of which holds it and the slipper in the main trunnion plate MTP, in which latter it is free to turn. trunnion plate is provided with a pair of smaller trunnion pins carried in bearings secured in their seatings in the cradle C.

The whole forms a sort of universal joint or gimbals, giving the slipper freedom for movement in two planes at right angles, corresponding to the face and sides of the shaft guide.

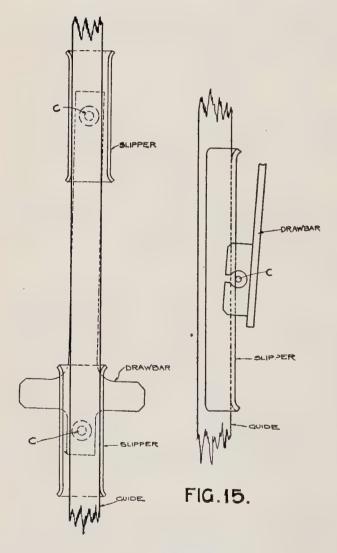
The cradle C, which may be part of the drawbar forging or merely riveted to the latter, is provided on its inner face with a steel rocker bar SRB against which the trunnion T constantly bears, thus relieving the small trunnions and their bearings from any direct guide pressures. The holes in the cradle in which the bearings are secured are open in front, so that, when worn, the slipper may be easily taken out and replaced while the skip is on the hoisting rope. This provision is shown in the two top slippers on the skip, the bearings being removed.

The bearings are of somewhat unconventional construction. The maximum movement of the slipper when at work resulting, as it does, in only a very small movement at the trunnions, the writer decided to use rubber bushes at these points. These are shown in details. SC is a steel container into which the rubber bush RB, cut larger than the hole, is pressed. The bushes are made rather long so that, when squeezed into their working positions by the bolts or study of the steel container, they effectively prevent water or other matter from reaching the trunnions, and, at the same time, provide that elasticity of bearing which is most desirable.

For limiting the movement of the slipper when out of the shaft a small limiting bracket and plate marked LB&P is shown.

Particular notice is directed to the reserve of strength permissible in the construction. The circular trunnion T not only provides for a large bearing surface in the joint, but allows of a more than adequate number of rivets to take any loads it may be subject to.

Figure 15 shows diagrammatically the action of the slipper, which, due to its freedom to adjust itself at any angle, presents the whole area of its wearing faces continuously to the guide. The load is thus distributed over an appreciable



Action of the Swivel Guide Slipper.

length between guide supports, the guide is given a greater reserve of strength, and its alternate bending and straightening is reduced to a minimum. The flexibility of the slipper enables it to ride over irregularities without shock or jamming.

The clearance between slipper and guide in this diagram

is purposely exaggerated.

In Figure 16, the channel or angle bars between the top and bottom slippers have been retained merely as stiffeners; the channel being wide enough to be entirely free from contact with the guide.

The manager of the Robinson Deep, Limited, kindly allowed the device to be tested in one of the compartments of the Chris shaft, and to him and to his staff the writer desires to express his great indebtedness for the help they have given him. The device has been running for over three months, and from the start has given entire satisfaction. The skips and cages for a new seven-compartment sub-vertical shaft on this property are being equipped with the swivel slipper, and those existing will also be brought up to date in this respect.

In general terms the following advantages have been proved in practice:

The wearing strips of the slipper require less frequent renewal, as they each last about twice as long as those on the rigid type. In addition, to compensate for the effect of a spinning rope causing the strips on one side to wear more rapidly than those on the other, the slipper itself may be reversed, thus giving it a still further lease of service.

The worn slipper may be quickly replaced while the skip is on the hoisting rope, thereby saving considerable time and avoiding hindrance to hoisting operations.

The rivets in the skip body and attached parts remain good and tight over a much longer period, due to reduced vibration, and the necessity of stopping hoisting to renew or tighten loose rivets is largely obviated.

Instead of being tapered, the wearing strips are worn straight and parallel, presenting an even surface to the guide and creating a distinct polish on the latter, instead of the usually furred and roughened surface. The bolts and joints in guides and timbering require considerably less attention, remaining, as might be expected, tight and sound over a much longer period. In fact, the better condition of guides and shaft timbering generally, resulting from the use of the swivel slipper, has been very noticeable.

At the lower levels of the shaft, where the noise of passing skips is most apparent, it has been observed that the skip carrying the swivel slippers makes practically no noise, a fact which indicates its much smoother running.

The device allows of some reduction of the clearance

that is required with the ordinary rigid slipper.

The foregoing are some of the advantages, actually observed, and it may be reasonably deduced that the hoisting rope, together with everything it supports, will benefit in reduction of shock, vibrations, and resulting shaft friction. The winding engine, too, will share in these advantages in steadier load and reduced steam consumption.

On the principle of 'Safety First', the swivel guide slipper is to be strongly recommended. Its flexibility, and the absence of any tendency to dig into the fixed guide, results in comparative immunity from costly accident due to the ripping out of shaft timbering.

A marked saving in working costs must result from the decreased wear and tear on guides and timbering, the less frequent renewal of slipper wearing strips, the comparative ease and rapidity with which slippers can be changed, the less frequent repair of rivets, the improved working conditions of rope, etc., and the comparative immunity from ripping of guides.

DEEP-LEVEL MINING AND HIGH TEMPERATURES†

AN ENQUIRY INTO CERTAIN CASES OF SUDDEN DEATH PRESUMABLY DUE TO HEAT STROKE, WITH A REPORT ON THE ASSOCIATED CONDITIONS.

By A. Mavrogordato* and H. Pirow** (Members, S. Af. Inst. Eng.)

(Toronto, Ont., Meeting, August 26th, 1927)

INTRODUCTION

I.—HEAT-STROKE

All the higher animals may be regarded from the point of view of over-heating as internal-combustion engines that should maintain their temperature approximately constant regardless of surrounding conditions and output of work.

In the case of man, this temperature is about 98°F. to 98.6°F., and well-being demands the maintenance of this temperature within comparatively narrow limits.

Section 1.—Heat-production in animals:

There is a minimal heat-production related to the internal activities of the body; otherwise heat-production varies pretty directly with the amount of work done.

The unit of heat-production may be taken as the *large* calorie, i.e., the amount of heat required to raise the temperature of one kilogram of water by one degree centigrade.

Resting heat-production is at about the rate of a calorie per kilo. body-weight per hour. The extremes may be taken as from one calorie to nine calories per hour. The average man, over 24 hours, averages about 125 calories per hour, or rather under two calories per kilo. body-weight.

[†]This paper was published in the Journal of the S. Af. Inst. Eng., XXV,

No. 7, 1927. *Research Fellow, S. Af. Institute of Medical Research, Johannesburg,

South Africa.

**Government Mining Engineer, Union of South Africa; lately Assistant Consulting Engineer, Union Corporation, Limited, Johannesburg.

The work of an underground native varies; a trammer may be doing four to five times as much external work as a boy on machines, and, during the shift, the mean heatproduction may be taken as from three to four calories per kilo. body-weight per hour.

This heat must be dissipated if the boy's temperature is to remain approximately constant.

Section 2.—Heat-dissipation in animals:

The ability to dissipate heat is, under ordinary conditions, very great, and any considerable rise in body temperature ('fever') is related to failure in heat-dissipation and not to excessive heat-production. This failure may be due to external or internal causes.

Animals dissipate most of the heat they produce by radiation, convection, conduction, and evaporation. The heat is produced in the muscles and internal organs and carried from them by the blood to the lungs and the surface of the body. In this context, the lungs and the surface of the body may be compared to the radiator of a motor car and the blood to the circulating water.

Temperature is controlled, as in an incubator, by a thermostat known as the 'heat-centre'. The heat-centre is a small area in the brain connected with the blood-vessels, respiratory muscles, and sweat-glands. When the temperature of the blood bathing it rises, the centre responds by dilating the superficial blood-vessels, increasing the ventilation of the lungs and, if that does not do, bringing the sweat-glands into action. Leaving out the sweat-glands for the moment, man may be said to be fitted with a radiator that expands and contracts according to his rate of heat-production.

With the clothed body, each unit of surface gives off heat at such a rate as to maintain the temperature of the *surface* at about 97.5°F.

With the surface at this temperature we get: armpit 98.2°F., mouth 98.7°F., and rectum 99.5°F., or thereabouts.

Loss of heat by radiation and conduction relates to the temperature of the surrounding air and the rate at which it is changed. The body loses heat by warming the air in contact with it to its own temperature.

As long as the air is below the temperature of the body, the more rapidly it is changed the more readily heat is lost.

Clothing obviously obstructs rate of change, whether the air be 'moving' or 'still'.

The higher animals maintain their temperature constant even when the temperature of the surrounding air is above it, and that is where the sweat-glands come in.

SWEATING AND EVAPORATION

Cooling by sweating is cooling by evaporation.

'Latent heat', the heat employed in changing a body's state without changing its temperature, is responsible for cooling by evaporation. To change the state of one gram of water (sweat) to vapour takes over half a large calorie.

It is not palpable sweating that cools, but the impalpable sweating in the form of vapour. One often sees the statement that such and such conditions enable people to work without sweating. This does not imply that sweating is injurious, but that palpable sweating is wasteful, as sweat lost as water does not cool. A man streaming with perspiration is only vapourising a portion of the water he mobilises at his surface because evaporation rate is not keeping up with rate of water supply. This process, pushed far enough, *is* harmful, unless the water and salts lost are replaced by drinking.

A vapour is that which is given off from the surface of a liquid or the moist surface of a solid at a temperature below its boiling point. Evaporation is always going on at the surface of water, and this evaporation is accelerated by any process that removes the vapour rapidly from the surface of the water and is retarded and finally ceases if the vapour is allowed to accumulate in a closed space. When this equilibrium state is reached, the space is said to be saturated with the vapour; the density of the vapour is then the maximum which can exist in the presence of the liquid at the temperature of the test, and its pressure is called the vapour pressure or saturation pressure.

The saturation pressure of a vapour depends only upon the temperature.

Boiling occurs when the vapour pressure of a liquid becomes equal to the external pressure. Water at the temperature of the surface of the body (97.7°F.) has a vapour pressure of about 45 mm. of mercury, and at sea-level, at a temperature of 212°F., it has a pressure of 760 mm. of mercury and 'boils'. The capacity of the air to hold vapour rises with the temperature, and its vapour pressure rises with the degree of saturation.

RELATIVE HUMIDITY

The ratio of the weight of water vapour contained in a given volume of air to the weight which this same volume would contain when fully saturated at the same temperature is known as the 'relative humidity'.

DEW POINT

The 'dew point' is the temperature to which the air must be cooled to become saturated.

Table 1 gives the grams of water per cubic metre and the associated vapour pressure of saturated air over a certain range of temperature:

Table 1*

Temper	rature	Vapour pressure in mm. of mercury	Grms. of water per cubic metre (saturated)
C.	F.	(saturated)	(Survey)
0	32	4.5	4.7
5	41	6.5	6.6
10	50	9.1	9.2
1 5	59	12.6	12.6
20	68	17.3	17.0
25	77	23.5	22.7
30	86	31.5	30.0
36.5	97.7	45.3	42.0
40	104	54.8	51.0
50	122	92.0	83.0

^{*}Physico-Chemical Tables. John Castell-Evans.

Section 3.—Heat-stroke and contributory causes:

The human body, like any other mass with a damp surface, can evaporate into any air at a temperature below its own, even if that air be saturated with moisture, because the air in immediate contact with the body is warmed and so becomes unsaturated and, therefore, able to carry more water vapour, and it can evaporate into air at temperatures above its own as long as that air is unsaturated.

Evaporation rate will depend upon the difference between the water content of air saturated at the temperature of the body and the water content of the surrounding air. Leonard Hill calls this difference the 'physiological saturation deficit'. Reference to the table shows that air at 32°F. holds up to 4.7 grams of water, while air at 97.7°F., surface of body temperature, holds up to 42 grams of water, giving a saturation deficit of over 37 grams. Air at 122°F. holds up to 83 grams of water, so, at a relative humidity of 50 per cent, there will be a saturation deficit of about 41 grams ($\frac{83}{2}$ = 41.5 and 83 - 42 = 41). The film of moisture on the surface of the body is warmed to the temperature of the air, giving it a saturation pressure of 92 mm. of mercury as against the air's vapour pressure of 41 mm. of mercury at a relative humidity of 50 per cent, with a temperature of 122°F. This is how temperature is maintained normal in the tropics and in the hot rooms of Turkish baths at temperatures far above 100°F. Figure 1 illustrates these points.

Cooling by sweating, then, is cooling by evaporation, and evaporation rate depends upon the vapour content of the surrounding air and the vapour pressure associated with the

temperature of the air.

The atmospheric pressure is the sum of the partial pressures of nitrogen, oxygen, carbon dioxide and aqueous vapour.

One must think of the percentage of aqueous vapour in the air and the associated partial pressure as one thinks of the percentage of carbonic acid in the air and the associated partial

pressure.

The sweat glands play the same part in bringing the blood in contact with the air, from the point of view of vapourexchange, as do the blood vessels of the lungs in gaseousexchange.

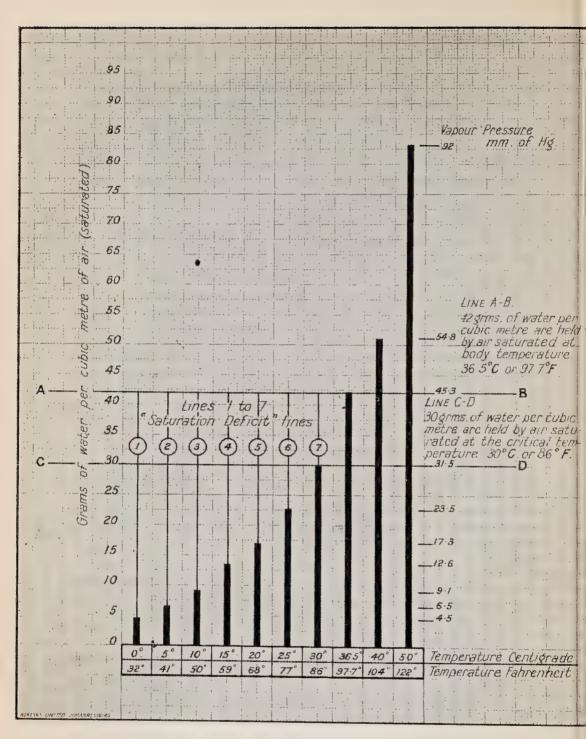


Figure 1

The black columns represent, roughly to scale, the amount of water per cubic metre he

by saturated air at the range of temperatures given.

The difference between the amount of water (42 grm.) that can be held by air at the temerature of the surface of the body (36.5°C.) and the amount that can be held by air at the difference temperatures given is the measure of the evaporative power of the air at those temperature and is represented on the diagram by the 'saturation deficit' lines 1 to 7. A fall in temperature saturation persisting or a fall in humidity temperature remaining constant will improve evaporation conditions and be represented by a lengthening of the saturation deficit line.

There is brisk vapour exchange in the lungs, and the expired air is always saturated with aqueous vapour. There is important heat dissipation by this route.

If air were found to contain three per cent carbonic acid, corresponding to a partial pressure of about 23 mm. of mercury, there would be consternation. Yet, carbonic acid in the blood is in equilibrium with gaseous carbon dioxide at a pressure of about 42 mm. of mercury, giving a pressure difference of 19 mm. of mercury. Saturated air at 86°F., a not uncommon condition underground, has about four per cent aqueous vapour at a partial pressure of, say, 31.5 mm. of mercury, while the tension (partial pressure) of aqueous vapour at body temperature is about 45.3 mm. of mercury, giving a pressure difference of only 13.8 mm. of mercury. (CO₂ at a partial pressure of 23 mm. of mercury in the helmets of divers is not uncommon, and they suffer no ill-effects from it.)

The facility of giving off aqueous vapour and the facility of giving off carbonic acid is conditioned by the same laws to the extent that it comes back to pressure difference.

WET AND DRY-BULB THERMOMETERS

The relative humidity can be determined by taking simultaneous readings of two exactly similar thermometers, one of which is kept dry — the dry bulb — while the bulb of the other is surrounded by wet muslin — the wet bulb.

If the atmosphere is saturated with moisture, then the readings of the two thermometers are identical; but, if the atmosphere is not saturated, the moisture surrounding the wet bulb evaporates, causing its temperature to fall. drier the air, the more rapid the loss of heat caused by evaporation and the greater the difference between the wet-bulb and dry-bulb reading.

Owing to the profuse use of water for dust-laying, the air of the Witwatersrand mines is almost saturated with moisture, and the wet and dry bulb thermometer readings come very close together. With the wet bulb reading of 86°F., the vapour pressure will be about 30 mm. of mercury, so the pressure difference will be only about 13.5 mm. of mercury. (See Figure 1).

Below 65°F., moist air feels cooler than dry air because its conductivity is raised, and up to this temperature evaporation is not an important factor in cooling.

While water, being a much better conductor than air, helps cooling by convection, it hinders cooling by radiation, as a rise in the amount of water vapour in the air renders air more opaque to the passage of radiant energy.

The distribution of heat-loss between radiation and conduction on the one hand, and evaporation on the other, depends upon the temperature of the surrounding air, its degree of saturation with moisture and the clothing. With air at or above 98°F., all cooling depends upon evaporation, and if the air be saturated with moisture at this temperature, the whole organisation of heat dissipation breaks down, and the body temperature rises even when at rest.

The ability to dissipate the extra heat produced by work begins to fall off long before a wet-bulb reading of 98°F. is reached, and, for the clothed European in still air, the critical temperature is round about 86°F. wet-bulb.

Heat stroke is due to external causes when the surrounding conditions make it impossible to dissipate the heat produced. In the case of illness, the heat-centre may be disordered, and, instead of keeping the temperature steady at about 98.4°F., it may either work irregularly or 'set' at a temperature considerably above normal. This is fever, due to internal causes.

At high temperatures, the wet-bulb readings are far more important than the dry-bulb readings, as they give a measure of the amount of heat that can be dissipated by evaporation.

They do not give all the information required, as they do not take into account the immensely important factor of air movement.

The air in two factories may have identical wet and drybulb thermometer readings and yet have entirely different cooling value if it be 'still' in the one case and 'moving' in the other.

It was said above that each unit of surface of the human body gives off heat at such a rate as to maintain the temperature of the surface at 97.7°F. If a sufficient rate of heat dissipation cannot be maintained, then over-heating occurs, and, if this goes far enough, heat stroke. The engine 'seizes' and stops.

To know the cooling power of air, we must know at what rate a unit of surface can give off heat to it, taking into consideration radiation, conduction, evaporation and airmovement. The instrument which gives this information is known as the 'kata-thermometer', and we owe this valuable instrument to Dr. Leonard Hill, who not only devised it, but, together with his school, has done an enormous amount of work with it under all sorts of conditions.

II.—THE KATA-THERMOMETER

Section 1.—The instrument:

The kata-thermometer is an alcohol thermometer with a range of 95°F. to 100°F.

This range is more or less arbitrarily chosen to give a mean temperature of 97.7°F.

In use, this instrument is heated to 100°F. and the time in seconds for the temperature to fall from 100°F. to 95°F, is noted.

The time taken will vary with the rate at which each unit of its surface can give off heat to the surrounding air. For each instrument a determination is made as to how long in seconds it takes to give off heat at the rate of one millicalorie (1/1,000,000 of a large calorie) per square centimetre of its surface.

This figure, having been determined, is known as the 'factor' of that particular kata, and is engraved upon it.

Suppose that when giving off heat at the rate of one millicalorie per square centimetre of its surface, it takes 476 seconds to fall from 100°F. to 95°F., then '476' is the factor of that particular kata.

If one is investigating cooling conditions with the kata, one notes the exact time it takes in seconds to fall from 100°F. to 95°F. and divides the figure found into kata-factor.

Supposing the mean of several determinations to be 95 seconds and the factor to be 476, the result is 5, near enough. This means that under the conditions studied the kata is losing heat at the rate of one millicalorie per square centimetre of its surface in 1/5th second.

The cooling value in this particular case is returned as '5', because the kata is losing heat at the rate of 5 millicalories per square centimetre of surface per second. This illustration has been chosen because a cooling value of '5' as determined by dry-kata has been found to be comfortable for rooms in England.

One must use 'wet' and 'dry' katas as one uses wet and dry bulb thermometers.

To convert the dry-kata into a wet-kata, one only has to cover the bulb with wet muslin.

The wet kata, being influenced by evaporation, will lose heat more rapidly than the dry-kata.

With a factor of 476, a wet-kata might take 30 seconds, on the average, to fall from 100°F. to 95°F., and 30 divided into 476 is 16, near enough. The instrument under the conditions studied has lost heat at the rate of one millicalorie per square centimetre of surface in 1/16th second. The cooling value in this particular case is returned as '16'.

The illustration is chosen because a cooling value of 15 to 16, as determined by wet-kata, has been found comfortable for rooms in England.

Katas being influenced by air movement will give different cooling values when thermometers are recording identical temperatures and, the temperature being known, rate of air movement can be determined from a simple formula. For this purpose one relies on the dry-kata.

At high temperatures the wet-kata determination is the more important determination.

Section 2.—The 'Kata-standard' of conditions:

For rough calculations one can take the average adult as weighing from 65 to 70 kilos, and having a surface of about 20,000 square centimetres. If such a one has a heat production rate of from three to four large calories per kilo. per hour, he must get rid of heat at the rate of about 3.5 millicalories per square centimetre of surface per second. Experience suggests that the wet-kata should lose heat at the rate of from four to five times the rate of the man if he is to work efficiently and

be comfortable. If the man is producing heat at the rate of from three to four large calories per kilo. per hour, then the wet-kata should read at from '15' to '17', at least.

Dr. A. J. Orenstein and the late Mr. H. J. Ireland, as the result of experiments carried out on the Village Deep mine,

reported

"When the cooling power of the atmosphere... is below, let us say, 16 units by wet-kata, the working efficiency of a native (stripped to the waist) falls off.

"In bad places, where the cooling power is only '5' wet or under, the average efficiency is only about 55 per cent, the body temperature rises to an undesirable degree and extreme fatigue may be produced by work."—(Journal Industrial Hygiene, Vol. IV., pp. 30-46, and pp. 70-91.)

In one of his earlier publications, Leonard Hill suggested certain cooling standards as judged by the dry and wet katathermometer, but he was at pains to add that these standards were tentative, and that an investigation should be made with a view to learning what conditions, as judged by the katas, actually obtained when health and output were satisfactory. Hill's tentative standards are often quoted without his qualification. The suggested standards for temperate climates were:

For sedentary workers: dry kata cooling power of 6, and wet kata cooling power of 18.

For light manual work: dry kata cooling power of 8, and wet kata cooling power of 25.

For heavy manual work: dry kata cooling power of 10, and wet kata cooling power of 30.

These standards are too high and, in practice, they are neither met with nor desired.

When one speaks of higher animals keeping their temperature approximately constant, one must accept a rise of 1.5 per cent to 2 per cent during sustained effort as physiological. Athletes, on hot days, frequently finish their game with a temperature of something over 99°F. and under 100°F. A similar rise of temperature has been found with mine boys at work and with Indian cotton mill operatives. This rise in temperature with effort is comparable to the rise of pulse of 10 per cent to 15 per cent that occurs with effort, and there

TABLE 2.
Section 2.—Table of Cases and Remarks.
Village Deep, Limited.
Deaths from Heat Apoplexy.

U			MAVROG	ORDATO AL	ND PIRO	W
	Remarks		Post Mortem —"Acute congestion of all organs."	Post Mortem—"heat apoplexy." Assaulted master and boss boy.	Post Mortem—"heat apoplexy."	Post Mortem — "Congestion of brain and edema of lungs."
	Was native a new boy or recently transferred to this working place?		Working first shift Post Mortem after being dis—"Acute concharged from City gestion of all Deep Hospital (o/a. organs."	Sub Incline Shaft cleaner A new boy working Post Mortem—"heat apopular incline Shaft. (night shift). first shift. —"heat apopular apopular incline Shaft. (night shift). Assaulted master and boss boy.	18/12/25 36 West Drive Trammer A shaft timber boy Post Mortem West Sub Incline (night shift) working second shift —"heat apolin this working plexy."	A new boy working Post Mortem second shift. "Congestion of brain and edema of lungs."
		Class of Work	Lashing	Shaft cleaner (night shift).	Trammer (night shift)	
	As per Report of Death	Working Place	6 W. 26	Bottom of West Sub Incline Shaft.	36 West Drive West Sub Incline Shaft.	1/1/26 Y. 27 West Stope. Lashing
		Date	29/9/25	15/12/25	18/12/25	1/1/26
		Time	:	3.30 a.m.	6.30 a.m.	11.45 a.m.
	Number of shifts worked		8	1	73	67
	Date en- gaged		31/7/25	Shangaan 11/12/25	15/9/25	F.C. 30/12/25
	Tribe		Shangaan 31/7/25		Xosa	
	ÖŽ	TAO.	338	316	2964	3588

The places where deaths had occurred were visited by us on days when the surface temperature was high and the following wet kata, 5.3. wet kata, 4.5. wet kata, 6.0. wet kata, 7.4. bulb, 90°F.; wet bulb, 89.5°F.; bulb, 88°F.; wet bulb, 86°F.; bulb, 87.25°F.; bulb, 87°F.; wet bulb, 86°F. records were taken:

is no cause for anxiety as long as 'normal' level or rate are resumed promptly when effort ceases. In the case of the temperature, as with the pulse, there are, of course, definite limits to the degree of rise that can be regarded as physiological. The temperature should not reach 100°F.

CASES OF HEAT-STROKE ON THE VILLAGE DEEP MINE Section 1.—Special contributory causes:

This enquiry has been called for owing to four cases of sudden death that occurred between September 29th, 1925, and January 1st, 1926. The summarized details are given in the attached table (Table 2), kindly supplied by the manager.

In discussing these cases, Mr. Tillard pointed to the following factors as contributory causes:

- 1. Owing to extraordinary hot weather prevailing during the period under review, it was found that the average air temperature at the bottom of the down-cast shaft, from which the whole mine is ventilated, was three degrees Fahrenheit higher than during any other similar period.
- 2. The recently-introduced practice of placing only one machine in a stope face reduced the amount of local cooling obtained from the release of compressed air in stopes.
- 3. Owing to the prevailing shortage of native labour, a number of the new recruits were physically not as robust as could be desired; in addition to this, the distribution of labour underground, with a view to allowing natives to become acclimatized before being employed in the hottest parts of the mine, was not as easily arranged as when a larger supply of native labour was available.
- Large engines and pumps recently installed underground increased the amount of heat which had to be dissipated.

We agree with Mr. Tillard that a combination of these occurrences contributed to the causation of the cases under review.

At the same time, it is obvious that the general underground conditions found at the great depth of workings must be regarded as the main factors, and that such conditions will be found on other mines when similar depths are attained.

For this reason a fairly detailed survey of the conditions was undertaken.

Remarks on Table 2:

There does not seem to be any evidence of race selection.

All four boys were working their first or second shift in a hot place. Two cases occurred on night-shift and two on day-shift, so difference in conditions between day and night does not appear to influence the incidence of heat stroke.

The post-mortem appearances were in all cases compatible with death from heat stroke, though, in the absence of the history, Dr. A. I. Girdwood would have been prepared to regard No. 3588 as a case of influenza, the lungs being quite compatible with such a diagnosis.

There was one case of 'delirium'. 'Fits' and 'delirium' are quite in keeping with heat stroke, and there are numerous references to such seizures in the old literature dealing with the Black Hole at Calcutta and with slave ships. There were many cases of heat stroke in the Mesopotamia campaign during the late war and similar attacks were often noted in connection with them.

Almost exactly this time the previous year, December 19th and 24th, 1924, to be accurate, there were two cases of sudden death underground. In the one case, acute encephalitis, and in the other, dysentery, were found post-mortem. Over the same period of 1923, 'fits' were of fairly common occurrence. (Government Mining Engineer's report, 1234/24.) The cases of acute encephalitis and dysentery may be compared with No. 3588 mentioned above.

One may assume that cases of heat collapse are not very uncommon in hot places during the summer months, though by no means all go on to a fatal termination. Reference will be made in the next section to the fact that the temperature exceeded 86°F, wet-bulb in all the working places where the present series of deaths occurred. (See Table 2.)

With regard to these cases, the following matters may be considered:

- (a) Seasonal occurrence.
- (b) Incidence on particular mines.
- (c) Case selection.
- (a) Seasonal occurrence.—Since underground temperature only varies by 3°F. between summer and winter, the recurrence of these cases during the summer months calls for comment. The explanation is probably as follows: It was pointed out in the Introduction that, for the European, a temperature of 86°F., wet-bulb, could be taken as a 'critical temperature', because it was about this level that he ceased to be able to maintain his temperature at the normal. While a rise of 3°F. is not ordinarily of much consequence, it becomes a serious matter when the temperature is already round about the critical level. The temperatures in all the working places where deaths had occurred were taken by us on a hot day and, in each case, they were above 86°F., wet-bulb.
- (b) Incidence on particular mines.—This, too, may be referred to the critical temperature factor. It is only those mines where temperatures of round about 86°F., wet-bulb, are experienced that will be affected by a variation of 3°F.
- (c) Case selection.—If heat-stroke be due to an alteration in conditions that affects large areas of a mine, and brings about an exposure that should affect similarly so many miners, why should so few have been taken and so many left? 'Gassing', following an explosion, does not behave in this comparatively benevolent fashion.

In this connection, the following points may be discussed:

- (a) The critical level.
- (b) Acclimatization.
- (c) Age and physique.
- (d) Self-protection.
- (e) Intercurrent maladies.
- (f) Rate of air-movement.

The critical level.—Because 86°F. wet-bulb is the mean critical level for Europeans, it does not follow that it may not be somewhat higher for natives. Should this be the case, only the less fit or more susceptible will be affected seriously until a slightly higher temperature is reached. T. Maloney (') came to the conclusion that the level was higher for coolies in Indian cotton mills, and the general standard of health among mine-boys, considered in connection with their output of work, suggests that it is higher here. It would be of interest to secure a series of reliable basal metabolism determinations on natives.

Acclimatization.—The outstanding importance of acclimatization in connection with exposure to high wet-bulb temperatures has general recognition, and reference to Table 2 shows that all the boys affected were working their first or second shifts under these conditions.

Age and physique.—Half-grown boys and poorly-developed men are more likely to be affected than strong adults.

Self-protection.—It was pointed out in the Introduction that output goes down with bad temperature conditions because the operative slows his rate of work, whether intentionally or unintentionally. We are informed that experienced shift bosses on the Witwatersrand recognise that a job of work for which they would detail, say, four boys in a cool part of the mine, demands five or six boys in a hot place.

It must be remembered, too, that underground work, even that done by natives, has to be learnt. At lashing, for instance, a beginner is going to take far more out of himself for a given piece of external work than is an experienced boy. The boss-boy's standard of work-speed is the speed of the experienced, and a beginner in a row of old hands will have difficulty in keeping up. Jobs should be learned in cool places, and it is neither kind nor wise to set a raw boy to learn lashing at 86°F. wet-bulb. We are aware that such things only happen in emergencies, but — are there many emergencies that justify them?

One may assume, then, that the boys soon learn to adjust their rate of work to the conditions, and it is the inexperienced who are most likely to overdo it, and the new-comers who are most likely to be hunted by boss-boys.

⁽¹⁾ Loc. cit., p. 38.

Co-existent maladies.—This may be a very important factor. It is held by medical men with a wide experience in the tropics that a large proportion of cases of heat stroke occur in those who are already suffering from a febrile attack, and that malaria may affect the stability of the heat centre. A sufferer from chronic malaria may lose, to some extent, his power of temperature regulation and control.

Mention has already been made of the fact that Dr. A. I. Girdwood would have been prepared to have called one of the cases of heat stroke, on which he performed the post-mortem, a case of influenza, apart from the history. An attack of influenza in the early stages may cause a considerable rise of temperature before it has time to produce changes in the lungs that would be recognisable post-mortem, particularly in connection with the congestion associated with heat stroke, and which may be due to it alone. Reference to two cases of sudden death that occurred in 1924 shows that one was found to have been suffering from acute encephalitis and one from dysentery. Both of these diseases are associated with 'fever'. The suggestion, then, is that if a man goes underground and attempts to work when his temperature to start with is at 100°F., or over, it is more likely to reach a dangerous level than if it starts from normal, particularly as his heat centre may be unstable. In the next section reference will be made to a small enquiry which does not lend much support to this view.

Air movement.—When dealing with high wet-bulb temperatures, cooling conditions will be influenced as much by rate of air-movement as by the temperature. In none of the places where cases of heat stroke occurred was air moving at a greater rate than 40 ft. per minute.

To summarise this section: the fact that only a small loss of life has occurred under conditions that might, theoretically, have produced a catastrophe of the order of that associated with gassing after a colliery explosion, may have been influenced by the following considerations:

- (1) The critical temperature level for natives may be higher than that for Europeans.
 - (2) It is the unacclimatized boys that are affected.

- (3) Some boys are less ready than others at adjusting their rate of work to conditions.
- (4) A coincident febrile attack may have been the deciding factor.
- (5) With two working places both at a dangerous temperature a difference in the rate of air-movement may make one safer than the other.

Section 3.—Temperature of boys working in hot places underground:

We felt it would be of interest to learn the temperatures of boys actually working at a temperature of 86°F. wet-bulb or above, and with the kind help of Mr. C. E. Deacon, native supervisor on the Village Deep, the following arrangements were made: Temperatures of boys were taken in the compound first thing on rising, while working at about mid-shift, and again on their return to the compound after work. All temperatures were taken in the mouth. We are aware that rectal temperatures are to be preferred for such an enquiry, but we are also aware that boys would regard such a proceeding as a familiarity.

Seventeen boys were taken altogether in two separate lots on different days and in different working places. The work was drilling and lashing.

The temperatures first thing in the morning were surprisingly low. With one exception, to be referred to directly, not one reached 98°F., no less than five were below 97°F. and one actually was below 96°F. This one was taken twice.

These low morning temperatures suggest a low basal metabolism.

It may be mentioned that the thermometers used were all tested.

The temperatures while working were very uniform. With the exception already mentioned, the lowest was 99.2°F. and the highest 99.6°F.

These temperatures are about the same as those found in athletes during violent exercise on a hot day on the surface, and those found by T. Maloney (1) in Indian cotton mill operatives while at work.

⁽¹⁾ Loc. cit., p. 38.

It appears that at the temperatures of the test, 86.4°F. and 87.2°F. wet-bulb, the temperature does not go up by so many degrees, but 'sets' at a new level about a degree Fahrenheit above the normal. While all reached a temperature of between 99.2° and 99.6°, they had had to rise a variable number of degrees to reach this level.

One of the writers was the subject of an experiment that has been performed fairly often with the same sort of result. A bicycle ergometer was ridden in a closed chamber at a temperature of 98°F. wet-bulb. The temperature rose steadily at the rate of about one degree F. in every three minutes, and when the trial was stopped after a quarter of an hour it was over 103°F, and apparently going well. It seems possible that there are two stages in the failure of temperature regulation when working at high wet-bulb temperatures: one when the temperature sets at a slightly higher level than normal, and a further one at which it rises steadily. It is perhaps more likely that the 'set' at a slightly higher level than normal is physiological, and that we did not reach the level at which the native's power of regulation begins to fail. We hope to secure a large number of these observations, and then we may be able to come to some conclusion. The above preliminary results seemed worth recording, as they suggest that the boys, as a whole, are not in serious danger at present. Reference was made to an exceptional case in the small series studied. One boy's temperature was found to be 101.4°F. He denied that he was in any way unwell, and related his temperature to the fact that he had spent a very pleasant Sunday. This, by the way, was Monday morning. As there was no evidence of illness, he was allowed to go down and was watched carefully. So far from the temperature rising dangerously, it had fallen to 101°F. by the end of the shift, and at the highest was just under 102°F. The boy was a cheery soul of middle age, and perhaps spent more of his time during the shift in meditating on past excesses than in working. Subsequent examination showed no evidence of disease; he was kept under observation for a week, and he remained perfectly well.

All the temperatures, including that of the boy just mentioned, were taken after they had had about an hour and a quarter to 'cool off'. The *rakehell's* temperature was then down to 99.8°F., and all the others were below 99°F. The lowest was 98.2°F., and the highest was 98.6°F.

Section 4.—Prevention of heat-stroke:

As far as possible, boys fresh to mining conditions should not be sent to work in the hot parts of the mines until they have had the opportunity of becoming acclimatized and of learning their work in cooler places.

Special attention might be given to the age and physique of the boys sent to work in hot places.

If practicable, boys going to work in places known to be about the critical temperature level might be inspected before going underground, with a view to picking out anyone who might be feeling unwell without being ill enough either to lay-off or to attract attention at a casual glance. When in doubt the pulse should be taken, and, if this is suggestive, the temperature should be taken, too.

Boys working in hot places are going to lose a great deal of water by sweating, and the water of sweat comes from the water of the body, and the supply must be kept up. Not only is water lost by sweating, but salt is lost too, and a certain proportion of salt in the circulating fluids is essential to the well-being of living organisms. Not only the water but also the salt should be replaced. It is the salt loss that causes cramp in stoke-holes and similar places.

Thirsty men do not want to drink salt water, but 'half normal saline', about 0.3 per cent sodium chloride, can be drunk, and there are other more palatable formulæ. It is apparently the case that salty drink loses much of its repugnance when one is suffering from acute salt shortage. Infants suffering from the collapse of summer diarrhæa take half-normal saline from the bottle with avidity, though they reject it with fury when well.

The boys should have every facility to get drinking water while at work, and it would be a further advantage if some salt-containing drink were available.

Section 5.—First-aid treatment of heat-stroke:

The following remarks refer only to first-aid underground: doctors will all have their own opinions and practice with regard to lumbar-puncture, venesection, infusion, iced-water enemata, wet-cradling and so forth.

The danger signal is the skin going dry. As long as a man is sweating freely he is fairly safe, and it is the exhaustion of sweating that precedes heat collapse. If a boy working in a hot place is noticed to have a dry skin, he should be watched carefully if he is not sent off.

If collapse occurs, bring about artificial sweating by sprinkling the body with water and blowing air over it.

Keep the skin moist until the boy is handed over to the doctor.

Remember that heat collapse is cardiac failure. The heart has been doing double work in its attempt to keep a large volume of blood flowing rapidly through the superficial vessels while maintaining an adequate supply to the working muscles.

Stimulant is indicated.

Survey of Underground Temperatures, Ventilation and Cooling Powers on the Village Deep Mine

Section 1.—Depths and rock temperatures:

To supplement the general information received from the manager, officials and employees, a series of underground inspections was undertaken, during which detailed observations of temperatures, cooling powers, etc., were made. The results obtained, together with information supplied by Mr. Ranson, the Mine Dust and Ventilation Official, are given in the tables below.

Table of Depths and Rock Temperatures (Kindly supplied by Mr. Ransome.)

(Killuly supplied by Mr. Kansome.)					
	Depth below	Rock temperatrre			
	collar	Degrees			
Level	In feet	Fahrenheit			
31	5,920	91.8			
32	6,068	92.6			
33	6,214	93.5			
34	6,374	94.2			
35	6,543	94.9			
36	6,704	95.6			
37	6,869	96.1			
38	7,032	approx. 97.0			

From these results it is clear that in a considerable portion of the workings the conditions obtaining present a serious problem from the point of view of future mining and health considerations. A comparison of the air and rock temperatures also shows that conditions would have been considerably worse but for the measures introduced by the management to reduce the temperatures.

In view of the fact that Messrs. Tillard and Ranson dealt with these matters very fully in a most valuable paper read before the Chemical, Metallurgical and Mining Society of South Africa in February, no attempt need be made to enlarge on the available information.

It may, however, serve a useful purpose to indicate here what steps have been taken by the management to effect improvements, and also to pass in review the main avenues which have been explored in dealing with high temperatures and low cooling powers in other parts of the world.

The main features of the measures introduced by the management, the details of which we were able to investigate by personal inspections and enquiries, are contained in a contribution for which we are indebted to Mr. Tillard. (See Section 3 below.)

Section 2.—Temperatures, kata-readings and rates of air-movement:
Since very full details from this point of view are given in the paper by Messrs. Ranson and Tillard referred to below, we are not illustrating these features at length. The following readings have been selected to bring out the conditions obtaining in the hottest parts of the mine in the absence of special devices.

Dry-bulb	Wet-bulb	Dry-kata	Wet-kata	Air movement ft. per min.
89.9	88.7	2.4	7.2	260
87	86	2.7	7.4	76
90	89.5		5.3	240
88.5	86.3		6.6	80
87.3.	85.4		8	65
90	87		4.6	130
81.8	80.8		8	40
90	88.8		5.5	160
89.7	89.0		5	
88.7	87.6		8	240
87.5	86.25	3.5	7	152
89	88	2.6	4.5	175
88	87.25	3.2	6	144
	89.9 87 90 88.5 87.3 90 81.8 90 89.7 88.7 87.5	89.9 88.7 87 86 90 89.5 88.5 86.3 87.3 85.4 90 87 81.8 80.8 90 88.8 89.7 89.0 88.7 87.6 87.5 86.25 89 88	89.9 88.7 2.4 87 86 2.7 90 89.5 88.5 86.3 87.3 85.4 90 87 81.8 80.8 90 88.8 89.7 89.0 88.7 87.6 87.5 86.25 3.5 89 88 2.6	89.9 88.7 2.4 7.2 87 86 2.7 7.4 90 89.5 5.3 88.5 86.3 6.6 87.3 85.4 8 90 87 4.6 81.8 80.8 8 90 88.8 5.5 89.7 89.0 5 88.7 87.6 8 87.5 86.25 3.5 7 89 88 2.6 4.5

In the above situations, although the management has secured volumes of air vastly in excess of those usually supplied to working places on the Witwatersrand, the cooling rates are still inadequate. Two matters are worth insistence: firstly, that the management have done everything in their power to relate air-supply to temperature, and, secondly, the comparatively disappointing results secured by moderate rates of air-movement when the temperature and relative humidity are really high.

In two situations where ice-cooled sprays were in use, the temperature of the air had been reduced to 84.4°F. and 82°F. wet-bulb from 88.7°F. and 87.2°F. wet-bulb, and wet-kata cooling rates of 11.7 and 12.6 has been secured; a very interesting result. As we wanted to learn what the mine could do, we were taken to a spot where no men were working and got the following readings: dry-bulb, 91°F.; wet-bulb, 90°F.; wetkata, 3.13.

As mines get deeper, managements are confronted with the problem of realising conditions compatible with health and efficiency, starting with a state of affairs such as the one just given.

Section 3.—Measures introduced by the management to improve conditions:

[Kindly supplied by Mr. Tillard.]

- Distribution of air. (1)
 - Provision of large volumes of air for lower levels. (a)
 - Provision of air for development ends (fans, doors, (b) brattices, etc.).
 - Provision of air for stope faces (closing off large (c) areas, system of pig styes and packs so arranged as to close off central portion of individual stopes, brattice cloth put up between individual pig styes and packs, blowers in stopes, etc.)
 - Provision of a clear return airway (31 level east to (d) 16 level and main fan).
- Cooling the air by the use of ice. (2)
 - In development ends.
 - (b) In return air winzes.

- (3) Distribution of native labour to provide for acclimatization of new recruits.
- (1) Distribution of air.
 - (a) In order to deliver a maximum quantity of air to the working faces of the mine, the upper levels are systematically closed off by means of air doors, and the main intake air is sent to the lower working levels, before splitting. At present the first split takes place at the 29th level, a vertical depth of 5,627 ft. from the surface. From this level the air is coursed to the working places, each working level receiving its quota of fresh air.

At the bottom ventilation split in the mine (at present 6,543 ft. vertical from the surface), a fan of 40,000 cu. ft. per minute capacity is installed. This fan is moved from level to level as through ventilation is established. It ensures a large volume of air being circulated in the lower development levels of the mine.

(b) The present practice of ventilating development ends is by means of electrically driven Schlotter blowers. The air is forced through 15-in. dia. galvanized iron pipes from the last through ventilation point to the face, the capacity of the blower being 3,500 cu. ft. per min.

When the length of conduit exceeds 500 ft., a 3/16-in. compressed air jet is added as a booster to the Schlotter blower, and the blower is moved forward on the level as additional through ventilation points are established.

A 25,000-cu.-ft.-per-minute capacity fan situated at the bottom ventilation split delivers air to the working shaft bottom through 30-in. dia. galvanized iron pipes. From this 30-in. dia. pipe, 15-in. dia. branch pipes lead to the development ends below the last through ventilation connection.

(c) For some considerable time the practice of sealing off worked-out areas has been carried out with a view to making the best possible use of the air

available. In individual stopes the air current is, as far as possible, coursed along the faces. As soon as a stope becomes opened out to any extent, closing off the central portion or back areas is commenced by sealing off the bays or openings in one line of packs on the strike near the bottom of the stope. In the passage ways or bays near the face where the 'lashing' of broken rock takes place, brattice cloth screens are hung; when 'lashing' or shovelling is taking place in any bay the screen is lifted, and dropped when no work is going on. By these means it is possible to get a maximum quantity of air passing up the faces where work is taking place.

If, for any reason, the quantity of air necessary to give a reasonable cooling rate is not obtained, compressed air boosters are installed.

- Owing to the fear of caving of the worked-out areas (d)in the top levels of the mine, the East and West sides are being equipped with a clear return airway These returns will connect the in the footwall. stoping areas of the mine directly with the main return airway leading to the fan. As greater depths are reached so will these returns be pushed ahead to maintain an open airway from the working places. The main return between 31 and 24 levels on the East side of the mine is nearing completion. From 24 level the return air will travel through stopes adjacent to the shaft pillar. It is thought that these stopes can readily be kept open.
- Cooling the air by the use of ice. (2)
 - For a number of weeks ice has been used to cool the air in development ends. The procedure has been to place ice on trays in the delivery end of the 15-in. dia. ventilation pipes.

By this means it is possible to reduce the temperature of the air being delivered from the mouth of the pipes by about 6°F. Even with the delivery end of the column as far as 15 ft. away from the face, the

temperature at the face behind the machines can be reduced by 2°F. to 3°F., giving an increase in the wet kata cooling rate of about 3 millicalories.

In one particular drive in which tests were made (at a vertical depth of 6,543 ft., and a rock temperature of 94.9°F.), ice in the ventilation pipe produced a wet-kata cooling rate of 9.9 millicalories per sq. cm. per sec. when the machines were not working, and 12.7 millicalories when they were working.

Experimenting with ice in winzes and raises is also being carried out by allowing the compressed air from the auxiliary ventilation pipe to impinge on ice suspended near the face; results show a wet-kata cooling rate of practically double that obtained without ice.

In addition, Venturi boosters are placed in the drive dead ends, and have the effect of agitating the air, thus assisting the Schlotter blower and giving an increased cooling rate.

(b) In a series of winzes chilled water is being sprayed into the warm air passing from the development section of the mine into the stoping areas. The mine water is cooled by means of ice in tanks placed at the tops of the winzes, from about 84°F. to approximately 45°F., and flows to atomizers placed at intervals of about 20 ft. in the various winzes.

The most recent figures show a dry-bulb reduction of 6.7°F. The air enters the bottom of these winzes at a temperature of 88.3°F. dry-bulb and 87.3°F. wet-bulb, and leaves the top at 81.6 dry-bulb and 81.6 wet-bulb.

The degree of cooling in these winzes appears to be a gradually improving figure, no doubt due to the fact that the chilled sprays and relatively cool moving air produce a more or less cooled zone of rock surface. By spraying the air in these return winzes the heat acquired in the development workings is almost eliminated, and the air temperature approaches that of the split air on this particular level.

Although only in the experimental stage, the cooling results obtained from the use of ice are distinctly encouraging.

Further measures contemplated by the management to improve conditions.

- (a) Provision of separate and distinct ventilation districts for the development and stoping section of the mine.
- (b) Provision of separate ventilation splits and returns for the three large engines on 31 level at a vertical depth of 5,920 ft. from the surface.
- (c) Provision for increasing the total quantity of air passing through the mine:
 - (i) Isolating the developing and stoping sections of the mine as far as ventilation is concerned is contemplated. When the scheme is completed, the warm air from the development workings will be overcast direct from the return airways, so as not to traverse the working stopes or come in contact with the air from the splits feeding these workings.
 - (ii) On 31 level, the beginning of the third stage of winding, at a vertical depth of 5,920 ft., are three large winding engines, which necessarily create a considerable amount of heat. The three engine chambers will be connected to a common return (which is now being cut) leading into the main returns to the fan.
 - (iii) It will become necessary, when by-passing the air from the engine chambers and development direct to the returns, to increase the quantity of air going down the mine, by installing a larger fan this is practically decided on.

(3) Distribution of native labour to provide for acclimatization of new recruits:

New boys are, as far as possible, placed in stopes where the conditions are most favourable as regards ventilation, or are placed in sweeping gangs from which the output per boy is not as great as from ordinary shovelling gangs. From these gangs when acclimatized they are drafted where required.

REMARKS ON MR. TILLARD'S CONTRIBUTION

In testifying to the efficacy of the measures introduced by the management, it may be as well to point out that much further improvement cannot be expected on these lines. It is clear that the majority of the measures can only affect individual localities or at most small sections of the mine, that the cost of further extensions will soon fall outside the economic limit, and, particularly, that bratticing and closing off areas cannot be carried out on a grand scale owing to the serious risk of increasing the total mine resistance to flow of air, and thus preventing the circulation of air necessary for dust and ventilation purposes.

In reviewing some measures adopted in other countries to deal with high temperatures we are therefore not attempting to put forward concrete suggestions, for, obviously, the limitations outlined above, together with limitations entailed in the nature of the measures themselves, apply. It is rather with a view to completing the list of avenues which have been or are being explored, as well as with the object of inviting a critical examination of such avenues from men who are able to investigate their practicability in various parts of the Witwatersrand, that this brief review is submitted. For it is obvious that, with the increasing depths of the workings and the rapid change of conditions found in mines on the Witwatersrand, these avenues will be explored and tested more assiduously than has been the case in the past.

It is, however, equally obvious that local conditions are so different from those obtaining in other parts of the world that some of the methods tabulated later will only be applicable in very modified forms — if applicable at all. The chief difference to which attention has frequently been drawn, but which may again be emphasized here, lies in the fact that in the Witwatersrand mines air containing a very high percentage of moisture has to be dealt with, and, in all probability, will always have to be dealt with. It is also evident that some of the measures are already being employed to-day.

THE RELATION BETWEEN COOLING CONDITIONS, EFFICIENCY, AND HEALTH, WITH A BRIEF COMPARISON BETWEEN COOLING CONDITIONS IN 'WET-PROCESS' INDUSTRIES ELSEWHERE AND CONDITIONS ON THE WITWATERSRAND MINES

Section 1.—A desirable atmosphere:

The Health of Munition Workers Committee defined a desirable atmosphere as follows:

Cool rather than hot.

Dry rather than damp.

Diverse in its temperature at different parts and at different times rather than uniform and monotonous.

Moving rather than still.

It will be seen that the deep-level mines on the Witwaters-rand, in the absence of mechanical ventilation, realise something like the ideally bad conditions. The air is hot, saturated with moisture, uniform in temperature and still. It is these conditions that managements have to combat. If one is to judge the conditions by the kata-thermometer, the local standard must be determined empirically because it is affected by nature of work, conditions of work, clothing and, very particularly, by acclimatization.

Section 2.—Some general considerations:

The removal of heat from the hot, moist surface of the rock of a mine will be influenced by the factors that affect the removal of heat from the hot, moist surface of the human body. The cooling effect of the air will be related to its temperature, its evaporative power, and its velocity. A low cooling power may be due to efficiency in any one of these three factors, and the choice of remedy will depend upon the predominant factor or factors concerned. This simple statement must be qualified by the recognition of the fact that atmospheric conditions in industry must be studied not only from the point of view of the comfort of the worker but also from that of the special demands of the process of manufacture.

In this context the Witwatersrand mines are comparable to the humidification rooms in the textile and rubber industries and to laundries, because the laying of dust by water brings about the high relative humidity that the process of manufacture demands in the textiles, etc.

A reference to the Factory Act will show how difficult is the problem raised in our case by 'process of manufacture'. The Factory Act does not permit artificial humidification at temperature above 75°F. wet-bulb, and only permits this temperature when there is at least five degrees difference between the wet and dry-bulb reading. In many cases we have to accept temperatures of 80°F. wet-bulb and over (up to 88°F. wet-bulb) and, owing to dust-laying by water, we often cannot get a difference of more than one to two degrees between wet and dry-bulb readings.

Poor cooling powers of the air on the Witwatersrand mines are related chiefly to the fact that air at a low evaporative power is moving at a low velocity; a condition similar to that of a humidification shed without special ventilating devices. The trouble in humidification sheds is remedied by cooling the in-coming air, by increasing the rate of movement of the air, or by a combination of these two methods. Cooling the air helps not merely by lowering the temperature but also because the evaporative power of the air is thus raised. The cooler the air, saturation remaining constant, the greater the 'physiological saturation deficit'.

Practice in America tends to cooling the in-coming air, in England to increasing the rate of movement without artificial cooling.

It must be remembered that it is not necessary to blow a gale through the shed; all that is required is a brisk movement around the operative.

A growing practice is to attach vanes to the rollers of mills and looms, while light canvas screens direct the air current on to the operative at close range. Local movement at the rate of 200 to 250 ft. per minute is readily secured.

It may be said then that our choice lies between general or local cooling of air, general or local increase in rate of movement, or some compromise between these methods.

As far as general methods are concerned it means great volumes of air at the temperature available or much smaller volumes of artificially-cooled air.

Table 3 Section 3.—Cooling conditions in some English and Indian factories

	Dry- bulb	Wet- bulb	Dry- kata	Wet- kata	Air move- ment (ft. per min.)
Cotton factories					
(England)					
No. 1	70.9	65.5	5.2	13.1	28
No. 2	70.7	65.8	5.4	15.8	31
No. 3	69.1	63.7	5.4	13.3	24
No. 4	71.6	64.6	4.9	16.3	25
No. 5	72.9	68.7	4.7	14.9	25
No. 6	65.5	62.2	6.2	17.7	26
No. 7	74.3	71.1	4.3	13.2	21
Effect of attaching vanes					
to loom:					0.1
Looms stopped	72.7	68.7	4.6	13.4	21
Looms working	68.0	64.6	6.0	17.7	33
With vanes	73.3	68.2	9.0	25.9	252
Rubber works	80.0	71.0	3.9	16.5	47
	85.0	74.0	2.8	13.3	47
Laundry	85.0	75.0	3.1	17.5	66
	90.0	78.0	2.3	20.0	141
Tobacco	81.0	74.0	3.8	15.6	52
Cotton factories					
(India)					
No. 1	85.4	80.6		9.1	
No. 2	84.0	79.7		9.6	
No. 3	89.8	83.5		8.9	
No. 4	90.0	81.8		9.2	
No. 5	91.3	84.6		7.2	
No. 6	93.8	85.5		6.7	

^{*}Out of doors anything up to 250 feet per minute counts as a 'light air' and does not move wind vanes. A 'gentle breeze' extends a light flag and may be up to 950 feet per minute. These velocities may be divided by two at the height of the body.

^{*(}Official Beaufort Scale quoted by L. Hill; Med. Res. Comm. Special Rept. Ser. No. 32, p. 53.)

As far as the health and comfort of the operative is concerned, there does not seem to be much to choose between the methods, and choice can fairly be based on costing.

Table No. 3 gives some data from other industries where humid air cannot be avoided. It may be of some interest to compare the conditions with our own. (Table No. 4 below.)

As far as the English factories are concerned, the Table gives some cooling conditions actually met with in great industries where process of manufacture introduces difficulties. They may be compared with tentative standards based on theoretical considerations.

A glance at the wet-bulb temperature shows that the factory problem is much simpler than ours, as both temperature and relative humidity are much lower.

If velocities of over 100 feet per minute be omitted, velocities which until quite recently were not at all typical, we find the average rate of air movement to be about 34 feet per minute, and the average wet kata cooling rate to be about '15'.

All observations are from readings taken close to operatives working in closed rooms or sheds and the air movement is secured by fans or other devices.

The 'bad' Indian cotton mills have been specially selected as illustrating conditions comparable to our own. Their state resulted in an 'enquiry', and the figures given are taken from the report (¹). The state of affairs, allowing for the natural temperature conditions, was found to be due to unintelligent methods of humidification and absence of ventilation. They have been much improved since the report was issued, and it is asserted that there has been a gain in output and a diminution in 'hours lost'.

Section 4.—Cooling rates, health and efficiency:

During the last dozen years there have been a large number of studies on output, fatigue and conditions in industry, and, among other things, it has been learned that, when other factors can be eliminated, there is a relation between output and cooling conditions. Man's instinct is to protect himself,

⁽¹⁾ Loc. cit. p. 38.

and Dr. John Haldane wrote long ago apropos hot mines in Cornwall.... "it is not the miner's health, but the owner's pocket that suffers."

Reference has already been made to studies on the Reef by Orenstein and Ireland. Reference may also be made to the fact that, in the Indian cotton mills, output always falls in the hot months. T. Maloney (1) writes:

"There is, however, ample evidence to show that when working conditions become uncomfortable owing to high wet-bulb temperature in combination with a high degree of saturation, the Indian mill operative to some extent accommodates himself to the more unpleasant conditions by a slower rate of work and more frequent absences in the mill compound. Thus, while body temperatures are not affected to the extent that might be anticipated, the effect of conditions is more appreciable if the output of the workman is considered.

"The very general practice among Bombay mill hands of returning to their up-country homes has a beneficial effect upon their general health as reflected by weight; and counteracts to a very large extent the effects of working and living conditions in Bombay."

Illustrations could be multiplied, but the above one has been selected as the general conditions of work in Indian cotton mills are comparable to our own in so many respects, e.g., high temperature, high relative humidity, most of the external work done by native labourers imported from upcountry homes and living in compounds.

The second quotation shows the importance of intermittent as opposed to continuous employment in a trying industry and bears out our experience with mining boys on the Rand.

Section 5.—Acclimatization:

In deciding as to what cooling conditions should be aimed at, one must never forget the fact of acclimatization.

T. Maloney, discussing Leonard Hill's tentative standards, writes:

⁽¹⁾ Loc. cit. p. 38.

"In tropical climates, the cooling power by dry kata is usually small, and in some cases negative, and kata values of the order given are generally impossible, by the ordinary methods of ventilation.

"From personal observations taken in Indian mills and workshops, a wet-kata cooling power of '11' is sufficient to prevent visible perspiration among mill operatives, but higher cooling powers up to '16' are advantageous.

"The reasons why a lower wet-kata standard would be suitable for Indian mill operatives are influenced by—

- (1) The lighter clothing worn.
- (2) The greater portion of the body exposed to the cooling influence of the air.
- (3) The lower basal metabolism of the Indian as compared with the English operative."

Anyone with experience of factories in England is often struck by the fact that although the shed feels hot and stuffy, yet the operatives are keeping many of the windows shut. H. M. Vernon refers to a pottery where the temperature was 80.4°F., but only 33 per cent of the windows were fully open, and writes: "The reason is that they come to the works at the age of fourteen, and they get so acclimatized to high temperature that they feel uncomfortable without it." (Medical Research Council, Special Report, No. 73, p. 81.)

If the health of the worker and the output be satisfactory, one must not be unduly influenced by the fact that conditions do not appear to correspond to theoretical standards.

Section 6.—Cooling conditions on the Witwatersrand mines:

There would be no difficulty in giving a number of instances from the Witwatersrand of conditions found in Indian cotton mills before ventilating devices were introduced.

It is most important to recognise this fact, but hardly necessary to illustrate it at length.

Table 4 is compiled from a large number of observations made up and down the Reef during the last five years. The readings have been deliberately selected to bring out certain facts:

(1) Despite our conditions, we do secure double-figure wet-kata readings in the majority of cases. If we omit velocities of over 100 ft. per min. as un-typical, and Nos. 13, 18 and 19, it is found that our average rate of air movement is about the same as that in English factories, while our wet-bulb temperatures average about 77°F. as compared with their 69°F. The result is that our wet-kata readings average just under 12, as compared with theirs of just over 15. No. 13 is omitted because the figure '5.8' for wet-kata is almost certainly a misprint, and Nos. 18 and 19 are omitted as picked 'hot-spots'.

Table 4
Some temperatures and corresponding air movements in mines on the
Witwatersrand

	Dry-bulb (F.)	Wet-bulb (F.)	Dry-kata	Wet-kata	Air move- ment (ft. per min.)
No. 1	76.5 80.0 77.0 79.5 79.2 77.5 79.7 82.0 79.7 79.2 77.0	74.2 76.5 8a 73.4	4.4 3.9 3.8 3.75 2.8 4.3 3.75 3.2 4.1 3.5 3.9 me after 9.6 3.3 me after 11.1	12.5 12.2 12.2 12.2 8.2 12.8 12.2 11.8 12.6 11.9 11.3 'holing throug 22.7 5.8 'holing throug 28.6	375 12 h' 530
No. 15	83.0	75.0 77.0 82.0 86.0 88.7	3.9 3.9 3.1 2.75 2.4	12.8 12.2 10.0 7.4 7.2	28 50 40 61 141

It must be admitted that we are not as good as the factories and that there is plenty of room for improvement.

(2) The condition varies with the rate of air-movement rather than with the temperature, and there does not seem to be sufficient attention given to relating air-movement to temperature. It is not uncommon to find working places comparable in almost every respect, except that 'B' is some 10°F. hotter than 'A'. Air-movement being about the same in each case, the result is that 'A' is tolerable and 'B' uncomfortable. Here is an instance, and instances could be multiplied: 'A'—Dry-bulb 75°F., wet-bulb 74.5°F., dry-kata 4.2, wet-kata

13.5, air movement 22.5 ft. per min.

'B'-Dry-bulb 86°F., wet-bulb 85°F., dry-kata 2.1, wet-kata 6.5, air movement 20 ft. per min.

Compare also Nos. 4 and 5 in the Table.

Nos. 11 and 12 and 13 and 14 are of interest as showing the temporary effect of 'holing through'. The readings were taken in stopes and on consecutive days in each case. This is the reason for including No. 13, although, as said above, the wet-kata is probably a misprint.

Nos. 18 and 19 are included as typical 'hot-spots', and bring out the difficulty of cooling by air movement alone when both the wet-bulb temperature and the relative humidity are really high. The Indian cotton mills have not our terrible relative humidities to cope with, and there is rarely less than 5°F. difference between wet and dry-bulb readings. In the case of No. 19, with a dry-bulb reading 89.9, it would require a velocity of about 550 ft. per min. to give a dry-kata of '4', and then it is unlikely that the wet-kata would exceed '12', as the wet-bulb reading is 88.7. It is under these conditions that artificial cooling has to be seriously considered. In both 18 and 19 an attempt has been made to relate air movement to temperature, the movement in the one case being about double and in the other about four times the average.

Wet-kata readings do not correspond directly with drykata readings, because they are influenced by the relative humidity.

In the mine observation all the air velocities are calculated from the dry-kata readings, and the kata, being influenced by eddies, does not give direct linear velocities. The factory observations, both English and Indian, are also calculated from the dry-kata, and are comparable. In the mines the air movement is not as constant as in a factory, and there may be a difference of 10 per cent one way or the other between the time of taking two sets of kata observations. Greater accuracy would be secured by taking dry and wet-kata readings simultaneously, but in most of the above cases they were taken in succession, there being only one observer.

From the cooling point of view, the kata, giving total air movement, furnishes more useful information than the anemometer, giving direct linear velocity.

Section 7.—Summary:

Experience on the Witwatersrand suggests that the wetkata should cool at from four to five times the rate a man's body has to cool if his temperature is to be maintained at approximately the normal level.

For a mine boy this works out at from '15' to '17' by wet-kata.

Orenstein and Ireland found in their studies on mine boys that efficiency began to fall off when the wet-kata cooling rate fell below '16'.

Maloney, in his report on Indian cotton mills, suggests a wet-kata cooling rate of '16' as advantageous.

The average wet-kata cooling rate in the British industries quoted works out at just over '15'.

It is of interest to note than these are about the figures originally suggested by Leonard Hill for sedentary workers. ('6' by dry-kata and '18' by wet-kata.)

One does not suppose that the well-being of the organism demands that the partial pressure of the oxygen of the air should be varied according to whether it is at rest or active just because there may be some evidence that a rise in partial pressure helps effort, and there is proof that any considerable fall hinders effort.

There is no doubt that any wide departure from normal in the direction of insufficient conditions of cooling hinders effort, but is there much evidence that increase in cooling facility beyond a point helps effort? It is the art of the living organism to maintain a close adjustment of the *milieu interne* in the teeth of wide variations in the *milieu interne*, and is it not perhaps a trifle unphysiological to expect anything so fundamental as 'cooling' to be dependent upon a close adjustment of external conditions to behaviour?

Observations over a period of thirty years at the Radcliffe Observatory, Oxford, have shown the mean rate of air movement out-of-doors to be the 'moderate breeze' of the metereologist. That is to say, an air movement at the height of the body of from just over 500 ft. to just over 700 ft. per minute. The higher velocity would give a wet-kata of about '16' at 81.5°F. wet-bulb.

The processes in living organisms — circulation, respiration, energy output and so forth — operate within very wide limits between rest and effort, and it is possible that spectator and player are equally well off at '18' wet-kata.

Our present better cooling figures on the Witwatersrand mines are about '12' wet-kata.

BRIEF REVIEW OF AVENUES FOR IMPROVING CONDITIONS

- 1. Cooling of down-cast air by means of refrigeration on the surface or underground.
 - (cf. Davies, Trans. I.M.M., Vol. LXIII, 1922.)
 - (cf, Clifford, Trans. S.A. Inst. of Eng., March, 1921).
- 2. Drying and cooling of compressed air used underground.
- 3. Cooling of air underground by means of ice or cold sprays (see also reference to cold drippers on page 600).
 - (a) in shafts and main air-ways;
 - (b) in individual development ends.
 - (cf. Tillard and Ranson, Jour. Chem., Met. and Min. Soc., February, 1926.)
- 4. Cooling of air underground by contact with cold water pipes from the surface.
 - (cf. Tillard and Ranson loc. cit., discussion.)

- 5. Maintaining the fresh air supply at a lower temperature than the surrounding rock:
 - (a) by means of lagging, or other forms of heat isolation, of pipes or airways in which the air is conducted;
 - (b) by means of increase of speed of travelling air;
 - (c) by admixture of (cool) compressed air.
 - 6. Cooling of the underground water supply:
 - (a) by means of cold water pipes from the surface;
 - (b) by means of ice transported underground;
 - (c) by means of water drawn from the upper levels.
 - 7. Distributing cool air to the working places:
 - (a) by splitting downcast air from main airways at suitable places and distributing it by the aid of fans and blowers along auxiliary airways or ventilation pipes;
 - (b) by closing off old stopes and other leakages by brattices, the use of a cement gun (cf. tests by Professor Dixon, City and Guilds Engineering College, London), caving or sand-filling abandoned stopes and, in conjunction therewith, providing airways clear of obstructions:
 - (c) alternatively, by providing a large volume of air to sweep around the working faces. This body of air to be kept at as low a temperature as possible by the means outlined above.
- 8. Cooling the rock surfaces exposed in permanent airways by means of the methods outlined above.
- 9. Adapting the lay-out and current development scheme to temperature requirements, *e.g.*, regulating the cross-sectional area of development ends, pushing on connections for through ventilation, running reef drives and footwall haulages in parallel, developing parallel auxiliary incline shafts with cross connections, providing distinct intake and return airways and laying out more definite ventilation districts where splitting of the air is practicable.
- 10. Adopting definite standards for temperatures and cooling powers underground.

- 11. Raising the standard of ventilation officials employed on the mines.
- 12. Providing for the carrying out of tests, experiments and research work.

SUGGESTIONS FOR IMPROVEMENT OF TEMPERATURE CONDITIONS IN HOT MINES ON THE WITWATERSRAND

One considers here four recognised methods of cooling hot working places:

- (1) General movement of air.
- (2) General cooling of air.
- (3) Local movement of air.
- (4) Local cooling of air.

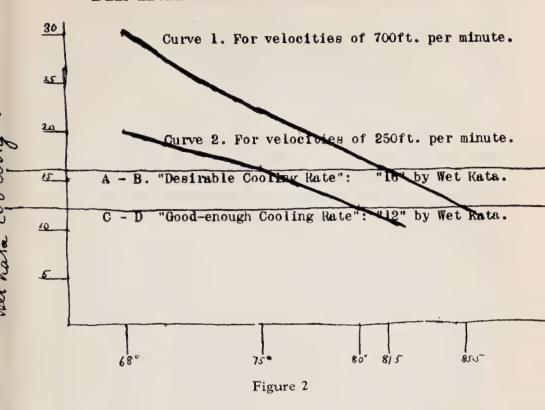
General Movement of Air.—This is limited by the volume that can be blown through the particular mine, and, allowing for thorough 'stopping' and saving as much air as possible for the hotter parts, it is probable that a velocity of 250 ft. per minute is the maximum that could be given to any considerable area of the workings. According to the late Mr. Ireland's calculations, a velocity of 250 ft. per minute gives the following wet-kata cooling rates:

68 °F. wet-bulb; '22' by wet-kata. 72.5 °F. wet-bulb; '18' by wet-kata. 77 °F. wet-bulb; '14' by wet-kata. 81.5 °F. wet-bulb; '11.5' by wet-kata. 86 °F. wet-bulb; '8.4' by wet-kata.

These calculated figures would have to be controlled by experiment for local conditions, and Mr. Ireland always insisted that they were no more than the most general indications.

It will be seen that, as far as these figures go, air of a temperature of 85.1°F., moving at 250 ft. per minute, gives a cooling rate of under '12' by wet-kata, and the amount of benefit to be secured by moving the air at this rate at a higher temperature would hardly justify a heavy expenditure.

An attempt is made to bring out this and some other points in Figure 2.



General Air Cooling.—In cooling mine air in bulk, as compared with cooling factory air in bulk, there are many differences, of which two are selected for mention.

- (a) Factories are fixtures, and a cooling plant that will serve a shed, or series of sheds, will have a long life in its chosen situation. In a mine, the men will be through a working place, or series of working places, in a twelve month. Unless the cooled air can travel a long way without heating up to the temperature of the mine air, this is a very serious consideration.
- (b) The mines on the Witwatersrand have the flattest temperature gradient of any in the world, because the rock gives off heat very rapidly. The gods give nothing for nothing, and the price paid for a flat temperature gradient is that the air heats up very rapidly on its way through the mine. If there is to be a really useful drop in the temperature, the air must be cooled in the immediate neighbourhood of the working place. With a shifting as opposed to a fixed working place, this is a great obstacle to the installation of any general cooling system.

If air below the natural temperature of the working place is to be used, it must travel, and travel fast, in heavily lagged pipes or brattices, if it has to go any distance, and this applies whether the air comes from the surface or a plant in the mine.

If it were decided to make the necessary excavations and instal a plant, experience on the Village Deep suggests that better results would be secured by cooling water and then conducting it in lagged pipes to be sprayed where it was wanted than by allowing cooled air to make its way through the mine to the working places.

Local Movement of Air.—The air-jet and the air-douche have gone far to solve the problem of cooling in humidity sheds, steel works, glass-blowing works, etc., but in a mine there are special difficulties. In a mine a large proportion of the men have not got fixed working spots during the shift and, since the air movement is only local, the outfit must follow them about. Another difficulty is that the rate of air movement that can be used is limited by the fact of working with naked lights. An ejector or fan will give almost any local rate, but, with naked lights, 700 ft. per minute is the limit that can be used.

According to Mr. Ireland's calculations, a velocity of 700 ft. per minute gives the following wet-kata cooling rates:

68°F. wet-bulb; '30' by wet-kata. 72.5°F. wet-bulb; '25' by wet-kata. 77°F. wet-bulb; '20' by wet-kata. 81.5°F. wet-bulb; '16' by wet-kata. 86°F. wet-bulb; '12' by wet-kata.

Local air movement is comparatively easy to secure, and up to 86°F. it is useful.

Local Air Cooling.—Air is cooled in humidity sheds by cold water sprays, and the effect of natural 'drippers' in mines is familiar. Where water can be cooled in bulk and taken in heavily lagged pipes to where it is wanted, the method is useful, as long as the water does not heat up unduly in transit. The experiments on the Village Deep show good local results with cooling water by ice on the spot and then spraying it, but this is hardly practicable on a large scale.

Summary.—It appears that local air movement on the jet or douche principle is the most helpful method, and in very hot places, above 86°F. wet-bulb, it might be reinforced by cold sprays. If enough air was available the same result would be secured by giving each man a compressed air jet to himself for cooling purposes.

Stopes are the great difficulty. It might be practicable to put some arrangement of the nature of 'rawl plugs' in the roof and suspend a canvas curtain from them along the line of the face. An ejector or portable fan blowing air between the curtain and the wall should improve conditions owing to the working party being in a sort of leaky brattice. The canvas curtain would only correspond to the portion of the face where the men were working and should move along with them. Its only purpose is to stop the local movement of air produced by ejector or fan being dissipated too quickly. Thus used it should not affect the equivalent orifice of the stope. (See Mr. Tillard's contribution.) Alternatively, a battery of ejectors might be used. After all, the volume of compressed air used by one drill would drive five ejectors.

Note.—This report deals with existing conditions and existing practice. Our temperature difficulties are due to the water used for laying the phthisis-producing dust but, while the proper use of water in an industry associated with a dust hazard always reduces the incidence of the associated malady and lengthens the safe working life, stone-cutters' rot has never been stopped by the use of water. If we can, without the use of water, retain the concentration of air-borne dust at the level at present secured by the use of water we shall possibly do away with miners' phthisis as far as the Witwatersrand is concerned, and, incidentally, solve our temperature problem. We have done some work along these lines and consider the matter worthy of further investigation.

APPENDIX

Note on the wet-kata standard as compared with the 'effective temperature scale' suggested by the American Society of Heating and Ventilating Engineers.

Since the above was written we have, thanks to the courtesy of Mr. C. P. Yaglou, Instructor in Ventilation and Lumination, Harvard School of Public Health, received a copy

of a paper summarizing some of the results secured by the Research Laboratory of the American Society of Heating and Ventilating Engineers in an enquiry on the thermal index of atmospheric conditions. The research has been in the direction of determining a scale of 'effective temperatures'. Allowing for the three factors — temperature, humidity and air movement — this school has produced a standard scale in which each degree can be made to correspond to wide differences in humidity and air movement. The procedure was briefly as follows: There were two chambers; in chamber number one the air was still and kept at a constant wet-bulb temperature, while in chamber number two the air was moved at various rates while both temperature and humidity were also varied. A large number of subjects — over one hundred — decided on the 'comfort vote' when chamber number two had thermal conditions sensibly indistinguishable from chamber one, e.g., "An atmospheric condition has an effective temperature of 65°F, when it is equivalent to a saturated one of 65°F, in still air." According to the scale they have worked out one gets the following figures:-

		Air	Effective
Dry-bulb	Wet-bulb	movement,	temperature
(F.)	(F.)	ft. per min.	(F.)
85.5	84.0°	Still	84.25°
		50	83.5°
		100	82.5°
		200	81.5°
		300	81.0°
		400	80.0°
		500	79.5°
		600	79.0°
		700	78 0°

Temperatures Fahrenheit (Wet-bulb) (See Figure 2)

Horizontal line A-B.—Desirable cooling rate: '16' by wet-kata.

Horizontal line C-D.—'Good-enough' cooling rate: '12' by wet-kata.

Curve 1.—Cooling rates secured with the air moving at 700 ft. per minute at the range of wet-bulb temperatures given.

Curve 2.—Cooling rates secured with the air moving at 250 ft. per minute at the range of wet-bulb temperatures given.

Air moving at 700 ft. per minute gives the desirable cooling rate up to a temperature of 81.5F°. wet-bulb and good

enough cooling rate up to 85.5°F. wet-bulb.

Air moving at 250 ft. per minute gives the desirable cooling rate up to a temperature of 75°F. wet-bulb, and goodenough cooling rate up to 80°F, wet-bulb.

N.B.—These curves do not pretend to be more than

general indications.

They have also worked out a 'comfort zone', that runs from 63°F, to 71°F, effective temperature. Within the comfort zone both comfort and output are optimal. In this comfort zone subjects are supposed to be clothed, and it is pointed out that in hot industries, with suitable clothing, or lack of clothing, and acclimatization, the comfort zone figures run higher.

If we compare the above figures with those given in this paper in terms of the wet-kata and based on Mr. Ireland's

Tables (see Figure 2), we get the following result:

At 84°F. wet-bulb, with air moving at 250 ft. per min., we get about 9.5 by wet-kata, while their 'effective temperature' would be just over 81°F. At 700 ft. per min. we should have 13.5 by wet-kata corresponding to their 78°F. effective tempera-For our conditions their upper limit of effective temperature for the optimal zone would be about 75°F.; that is to say, in the instance given we are three degrees effectivetemperature too high. By our methods we are 2.5 wet-kata units below the minimal optimal suggested (16 by wet-kata).

It is of interest to note that such very similar conclusions should have been arrived at by different lines of enquiry.

ACKNOWLEDGMENTS

In conclusion, we wish to express our sincere appreciation of the courtesy and assistance received from the Manager, the Mine Dust and Ventilation Official, the Native Supervisor, and all other officials and employees with whom we came in contact.

INTRODUCTORY REMARKS AND DISCUSSION

The Chairman (Mr. T. W. Gibson, Deputy Minister of Mines for Ontario): This paper should be one of considerable interest to us in Canada, especially in Ontario. We have not yet reached deep mining, as it is found in South Africa. The deepest shaft we have, I think, is the one at the McIntyre gold mine, sunk to a depth of a little over 4,000 feet. But we are on the way. We will be mining at deep levels, no doubt, in the near future, and we would like to take advantage of the experience of our South African fellow miners. We want to know what problems they met in deep mining and how they dealt with them.

As far as temperature is concerned, I do not know that this has as yet given us any difficulty in Ontario. Temperatures do not appear to increase greatly in depth as we descend. That condition may alter as we get farther down. If either Mr. Mavrogordato or Mr. Pirow is present, will he kindly present this paper?

I am informed that these gentlemen are absent, and in their absence Mr. C. J. Gray, Chief Inspector of Mines, South Africa, has undertaken to take their place.

MR. C. J. GRAY (South Africa): I do not propose to review this paper at any length, but I may tell you how the paper originated and indicate a few points which appear to me most likely to be of interest to members of the Congress.

Dr. Pirow, who is now Government Mining Engineer for the Union of South Africa was, nearly two years ago, a member of the staff of the Mines Department. The Department was concerned with regard to deaths which occurred in some of the deeper mines, particularly in the Village Deep mine, and decided that investigation should be made in addition to what was being done by the mine managers. Sir Robert Kotzé who was then Government Mining Engineer, arranged that the work should be done by Dr. Pirow as an Inspector, and Dr. Mavrogordato a medical officer attached to the South African Institute of Medical Research, who had devoted many years to the study of mine diseases. They studied conditions regarding temperature and ventilation in the Village Deep

mine and prepared a report which was submitted to the Department of Mines, though at the time it was completed Dr. Pirow had resigned from the Department. The paper is mainly a summary, with slight alterations, of that report, though the paper is based also on the circumstances of certain deaths which occurred in other deep mines with regard to which the authors had information.

There is information now available regarding several additional deaths from heat stroke which occurred either previous or subsequent to that report. All those cases bear out the conclusions which have been drawn in the paper and do not show any feature which would conflict with the views

there expressed.

One point of interest is that all the fatal cases are among the natives. So far as I know there have been no fatal cases of heat stroke among the white employees of the mines. is possible that the difference in the constitution of the white man and the native would explain that, but it is more probably due to the difference in work. The native in South Africa does most of the manual labour in the mines, the actual shovelling, the actual pushing of the trams, and the actual drilling where that is done by hand labour and not by machines. It may be that a white man, feeling that he is about to be overcome, gets to a safer and cooler place.

Another striking point is that all these deaths were of natives who were either new to the mines or working their first or second shift after returning to the mine, or of natives who had just been transferred to hot stopes. There are no cases of fatal heat-stroke among natives who have worked in hot stopes for any considerable period. On account of that, the practice has originated in the deeper mines of putting new boys, new natives, into training gangs. They are put on comparatively light work under the supervision of particular 'boss boys' who are instructed to see that they do not do too much work for the first week or ten days. After natives have gone through those gangs they do not suffer from heat stroke. It seems an extraordinarily large degree of acclimatization can be obtained.

Another point which would seem to be of interest is that the gradient of rock temperature on the Witwatersrand is

very low. It is approximately one degree Fahrenheit for 212 feet of actual depth. That is a lower gradient than the airtemperature gradient due to the adiabatic compression of air. If you have an air current going into a deep mine it will increase in temperature owing to the increase in pressure at the rate of about one degree for 183 feet. Thus you do not cool the mine by passing air down a shaft, if that air remains dry. There has been a good deal of discussion with regard to the effect of moisture in the air. We all recognize that humid air is more trying than dry air, but it has to be remembered that if we do not have the evaporation of water which produces that humidity we shall have a higher temperature. It is true that dry, hot air may be less objectionable than relatively cool, humid air, but considerations of temperature and humidity must be balanced. We cannot deal with one aspect of the matter without remembering the other.

Conditions differ in different mines, and though there are mines with higher temperatures than those on the Rand, they do not all, so far as I know, have the same difficulties. If the members of the Congress who have had experience in those hotter conditions can tell us how they have combatted them and overcome them, I am sure that information will be valued in South Africa.

The Hon. F. W. Beyers (Minister of Mines, Union of South Africa): I am not a technical man, but I simply want to say that I remember, as an inhabitant of Johannesburg, the time when deep levels were pooh-poohed—not only by the general community, but also by the mining men, the practical mining men, and by professional and technical men. Luckily, that has been falsified. If it were not for the deep levels what would the Witwatersrand have signified? Comparatively little, compared to what it means, and has meant, to the Transvaal and the world at large.

Professor K. Neville Moss (England): One of the most difficult problems of deep and hot mining is to ventilate adequately certain outlying workings in extensive underground operations. Such isolated, abnormally hot places call for physical knowledge of air conditioning, information as to the physiological effects upon workers therein, and a wise applica-

tion of this knowledge to practical conditions in the mine. The paper by Drs. Mavrogordato and Pirow gives a good summary of much of the existing knowledge on the subject.

I have the honour to be in charge of the Mining Department of the University of Birmingham, in which a good deal of work has been carried out for the Hot and Deep Mines Temperature Committee (a committee of the Institution of Mining Engineers, of England). I propose to give you a brief summary of the work which has been done since the first meeting of the Empire Congress in London.

I think I was one of the first to point out the importance of acclimatization to heat. I first came in contact with this at Pendleton colliery in Lancashire. There the underground temperatures in the deepest workings vary from 100° to 105° dry bulb, and from 85° to 93° wet bulb. These conditions are comparable with those obtaining in the deepest South African mines.

My first piece of work was to determine the amount of body loss by sweating, and there I found that one man in particular lost eighteen and a half pounds by sweating during five and a half hours work. That is a tremendous amount to lose. I don't want you to get the impression that that was an actual loss in body weight. During the shift in the mine the man drank a gallon of water, which is ten pounds, so that he actually lost the ten pounds of water which he drank and eight and a half pounds of water from body tissue. Nevertheless, he lost by sweating, even after making an allowance for moisture loss by breathing, eighteen and a half pounds in five and a half hours.

Whilst investigating the sweating problem, I came in contact with muscular cramp, very severe muscular cramp, which laid out the men. We discovered, after a certain amount of work, that it was due to the lack of chlorides in the body, produced by sweating and the excessive drinking of water. Kidney excretion in one particular instance gave no sign at all of chlorides, which is almost unheard of physiologically; and that one particular determination enabled us to realize what was happening. The men were definitely short of chlorides.

I then brought a collier down from Pendleton to the University of Birmingham and experimented upon him. We made him do work in a chamber heated to the temperature of the mine. His work output, measured by means of a cycle ergometer, was equal to his output of work in the mine. The output of work during actual mining operations had previously been determined. Then we collected all the salt he excreted and found that he lost about ten grams of salt in five and a half hours.

The thing to do was now obvious; it was to put ten grams of salt back into the men liable to cramp, by placing it in their drinking water. This we did and so overcame entirely the very bad attacks of cramp. Further, we very much delayed the period of fatigue which accompanied work in hot mines.

The introducer of this paper referred to heat stroke. I do not know of a single case of heat stroke in a mine in England, except in connection with Mine Rescue work. We do not get heat stroke in England, simply because the mines become hot gradually, and we do not put an unacclimatized man into hot places. That is the reason why you get heat stroke in South Africa. If you allow a man to become acclimatized before you put him into a hot place, he will not get heat stroke. Some of your natives have evidently been put to work for the first time in a hot place with the result that some have had heat stroke. What we get is, very occasional cramp, and more frequently severe fatigue. We have overcome the former, and, to a certain extent, the latter, by giving men salt water to drink.

I stated in a paper which I read about four years ago before the Royal Society, and also before the Institution of Mining Engineers, that the salt which is put into the drinking water should be made up of sixty per cent of sodium chloride and forty per cent of potassium chloride. That statement was made on a determination carried out on one individual collier. Since then we have done much more work on that subject, and we find that on the whole the salt of the sweat contains only about ten per cent of potassium, so that in putting back the salt into the body one would now suggest ten per cent

potassium chloride and ninety per cent of sodium chloride. The potassium doesn't appear to matter so much; sodium chloride appears just as effective.

Just recently I have been carrying out experiments on native students from India in order to see if it were possible to find out how these people could stand temperatures better than the Europeans, if in fact they could. From the observations which I made on these students I found that the salt concentration of their sweat was considerably higher than that of the European. This being so, they wouldn't suffer from water poisoning, which produces heat exhaustion, to the same extent as a European, because there would not be the same inducement to drink.

I will explain this a little more clearly, gentlemen. Suppose you take a long 'cycle ride on a very hot day and don't drink; you have a thirst, due to the abnormal salt concentration of your blood, and you quench that thirst by drinking. You quench that thirst, of course, to bring back to normal the abnormally high concentration of your blood. On the other hand, if you drink copiously of salt-free water, drink too much of it, the salt concentration of the blood becomes subnormal, and under such conditions body fatigue very quickly sets in. The salt concentration of the sweat from Indian students was from 0.35 to 0.4, whereas that of a European averages 0.25. Thus sweating would not produce so high an abnormal concentration of the blood in natives of India, symptoms of thirst would be less pronounced, and, in consequence, there would not be the same danger of drinking to excess.

I said that coal miners lose a good deal of water through the skin by sweating. One particular miner, working under extreme conditions in a hot chamber, lost six and a half pounds in one hour. That, of course, is very excessive. A theoretical calculation shows that a miner should lose by sweating, under ordinary working conditions in a hot mine, about six and a half pounds during a shift of seven hours. That is just sufficient to keep the body temperature normal. Sweating much in excess of that is mere waste, from the body-cooling point of view.

The acclimatized man wastes his sweat. Why he does, one doesn't know as yet, but he does. Excessive sweating will, of course, follow from excessive drinking due to an attempt to alleviate the parched feeling at the back of the throat. An unacclimatized man entering the mine will sweat profusely for an hour or two; then the sweat glands begin to dry up, and if he continues to work he will have a heat stroke. An acclimatized man can sweat continuously throughout the shift, keep his body temperature normal, and work without ill effects.

As to the cooling power of the air, that of course is very important. As pointed out in the paper, it is very essential. The recent work we carried out showed that the cooling power of the air is proportional to the square root of its velocity. Hence, a very small movement of the air will have a very appreciable effect. Air moving at 120 feet per minute gives double the cooling power of still air, and the effect is much greater than that due to an increase in velocity from 200 to 320 feet per minute. The cooling power at high velocities is not so great, proportionally, as that of low velocities.

It is stated, in this paper, that heat exhaustion primarily affects men who are not physically fit. With that I agree. I pointed out in 1923 that many of the men who suffered from severe abdominal cramp were men of poor physique and in poor health.

Dr. Leonard Hill drew up standards for the wet and dry kata thermometer and laid them down as being applicable to mining. Every mining man knows they are not applicable to mining. At Pendleton I found a set of men working in a stall in which the wet-kata cooling power in milli-calories per square centimeter per second was only four. They were turning out more coal from that stall than any other stall in the mine, and their absenteeism was a good deal less than the average for the pit. Strong, physically fit men, get acclimatized to conditions which the average person couldn't stand up against for more than half an hour. The importance of acclimatization cannot be stressed too much.

The only other thing I should like to mention, ladies and gentlemen, is that I have never seen (and I have followed the work in South Africa pretty closely) any statement in any of your papers on the heat of oxidation in your mines. You

have done a good deal of work on air movement and hygrometric conditions, but do you know for certain where the heat in the mines is coming from? Is some of the heat due to oxidation? You must have a great deal of timber in your mines, and oxidation of timber will produce a good deal of heat. The oxidation of timber will produce about 483 B.t.u's per cubic foot of oxygen absorbed. What about the oxidation of pyrite?

If you analyse your intake and return air you can determine the oxygen absorption and thus the heat production by oxidation. If any of you gentlemen have figures on that point, I should be very interested to hear of them. It would, to a great extent, help one to understand more clearly your

problem.

MR. V. B. REICHWALD (England): I have been very much wrapped up in the metallurgical industries, including glass manufacture. Some of the remarks that Professor Moss has just made have called to my mind experiments that were carried out in a Continental glass factory for some years, which seemed to run rather parallel with the experience you

have in the deep-level mines.

At that particular works they employed three glass tanks for smelting glass in large volume. The tanks would hold up to one hundred tons of molten glass, and this was removed from the mouth of the tank by large apertures, through which high temperatures were, of course, reverberated. The temperature reached up to 900°C. at the mouth of the tap-hole. The men working there were stripped to the waist, wearing only blue linen trousers. They were subject to two or three different effects of temperature.

Firstly, there was the radiated temperature from the open tap of the tank; secondly, the temperature that arose from the swinging pit of the glass; and thirdly, the body temperature raised by blowing the glass ball into a cylinder

by mouth blowing.

Your mine-temperature rise, as far as I am informed, is due to the stray minute particles of metallic matter. The consumption which is so prevalent among glass blowers is due to the strain from the exhaust air and the intake air upon the capillary tubes of the lungs. The trouble they encountered in glass blowing from the large tanks was that large volumes of glass were handled, and the men had not sufficient body moisture to stand the strain. There were cases of abdominal cramp the same as you have heard described, and also muscular cramp of the arms.

Experiments were carried out at this factory to ascertain what was the best thing to give the men to raise their powers of perspiration, i.e., their yield of perspiration. Experiments were carried out with three different liquids. The first was with a light alcoholic beer, which these men had previously been drinking when working. What the advantage of light alcoholic beverages of that sort would be beyond inducing the formation of plenty of saliva, which is necessary for the mouth, I cannot say; but for the tank work light alcoholic liquor failed.

The second set of experiments was carried out with a mixture of lemon juice and orange juice, unsweetened. Sugar would have raised the body temperature. Instead, salt was added, and that is comparable with what we have just heard, that they gave the men in the mines salt water. The lemon and orange were taken in a proportion of three to two. That was good so far as it went, but in the course of time the medical officer at the works ascertained that the large amount of lemon tended to reduce the consistency of the blood. In other words, the men got thin blood.

The third test was carried through on coffee and hot milk, also not sweetened with sugar, but salted. It was ascertained after experiments, continued over the best part of a year, that the men sweat best and sweat normally on copious quantities of coffee and milk: the coffee giving sufficient stimulation; the milk, with its lactic properties, giving sufficient feeding capacity to the bowels; and the salt replacing the loss of the chlorides, which has been referred to in the paper.

It was ascertained that, on the average, thirty per cent of coffee (pure coffee, without chicory) and seventy per cent of milk is the correct mixture, and it is used in those works at the present time, available for the men at all times of the shift, day and night, summer and winter. They don't change the beverage in the summer.

Coffee and milk is provided in such quantities that each blower is entitled to two cups of coffee and milk after each spell, which takes about twenty minutes of continuous work.

MR. J. McEvoy (Canada): I have listened with a great deal of interest to the discussion, especially that regarding the experiments to overcome the effects of heat in deep mines, and if you will pardon me for making the suggestion, it seems as if you are concentrating most of your brain power upon the problem of curing the disease after it is contracted, instead of stopping it before it gets started.

MacAdam gave a rule with regard to road building. It was: first drainage; second drainage; third drainage. The same rule might be paraphrased for the mines, as a remedy for the physical disabilities experienced in deep mining: first

ventilation; second ventilation; third ventilation.

I do not know to what extent experiments have been carried on to show what reduction in temperatures will be accomplished by increasing the air currents, but I am quite convinced that in the Pendleton colliery, when I visited it some years ago, the ventilation was totally inadequate, from the point of view of cooling the air in the mine. It was scarcely perceptible at the foot of the shaft, where the temperature was already ninety-six degrees. That was the bottom of the direct intake air current. When you have a good current of air going through you have a cool mine. You have the refreshed, clear air working. The air is not directed to the working faces. It certainly was not directed to the working of the long-wall faces in the Pendleton mine. The men were working entirely naked with the exception of a little leather skull cap. It was insufferably hot. The air did not circulate.

I think the same may be found to be true of the metal mines, where the law does not compel any particular quantity of air. In gaseous coal mines, they are compelled to have a certain amount of air and, as a rule, in the coal mines in this country, they put through the mine over five times the amount of air, sometimes ten times the amount of air, which the law prescribes. Consequently, they have good conditions for

working.

I throw out the hint as to whether you could not stop the disease rather than try to cure it after it is started.

PROFESSOR Moss: I would like to reply to that. In Pendleton, which is a very old colliery, the shaft is of small diameter. It is impossible to get much air down that shaft, and they are working at considerable distances from the pit bottom.

I agree that the heat conditions down there are very severe. One has to more or less doctor the men because one cannot doctor the mine. I quite agree that the only way of tackling the problem of heat in mines is to put as much air as possible into them.

As far as I can see, we shall, in the future, have to concentrate our work and our air in one particular section of the mine. We must get air down in greater quantities. We want large quantities of air and air movement to keep the men cool.

MR. C. J. GRAY: May I refer to the matter from the South African standpoint? I think the paper clearly shows that the importance of ventilation is recognized by the authors. They show that a great deal has been done and that a great deal is desirable. The difficulty, of course, is to get a larger quantity of air through a given opening. There is a limit to what is practicable, and we have the outstanding difficulty that if we pass air down to a great depth we increase the temperature of that air by compression. Therefore, we cannot get cool air in depth unless we have some artificial means, either by the evaporation of water, or by other cooling methods, of reducing the temperature. It is a practical difficulty which we are up against.

MR. F. W. GRAY (Canada): Mr. McEvoy has raised a point that is of very considerable interest to us in Nova Scotia, where our mines are very largely submarine. We are very much interested in the matter of ventilation in regard to cooling in the mines of Nova Scotia.

Generally speaking, we have not yet, in Canada, been troubled with any high temperatures. Our mines are comparatively shallow and our deepest metal mine, I think, is little over 4,000 feet. In Nova Scotia we have not as yet anything that is deeper than 1,100 or 1,200 feet. However, we have mines that extend under the sea, and in one instance we are now about two and a quarter miles under the sea.

We find that the question of ventilation is going to supersede all other considerations in regard to haulage and transportation. The temperature gradient and the strata are apparently cooler than in Great Britain, because we have mines that are 1,500 feet below the level of the ocean and over two miles out to sea and yet we do not have face-temperatures that exceed sixty-three or seventy degrees Fahrenheit. I have never seen a man underground in the Nova Scotia mines who took his shirt off. They don't divest themselves of their shirts, as they have to do in the deep Lancashire mines. If a man is temporarily idle he has to put his coat on, otherwise he gets chilled.

In studying the question of the future of these submarine mines, we have concluded that ventilation is going to supersede all other considerations, and in one instance we propose that we may have to pass something like a million cubic feet of air per minute in order to get the necessary amount of ventilation through the long circuit it will have to travel. There is no chance of relief in a submarine mine, and the air has to travel in and out of virtually the same surface opening.

The matter of sweat concentration is rather interesting—not exactly in the matter of heat, but we have had a great deal of trouble in some of our collieries with electric safety-lamps equipped with batteries carried on the hip. In the case of some men we positively could not keep the battery cases intact. The sweat is of such an acid character that it eats away the metal and we have had to use extremely heavy nickel plating for the battery cases.

I don't know whether it is a matter of interest or not, but we find that the Hungarians are the people who are the worst in this respect. Men of other nations don't seem to bother us. Whether it is a matter of diet or not, I don't know.

SIR RICHARD REDMAYNE (England): I am rather loath at this late hour to add anything in this most interesting discussion, but this is such an interesting subject and of such paramount importance to us in the old country in regard to our coal mines, which are becoming deeper and deeper, that I do venture to offer a few remarks.

About six weeks ago I studied the conditions at the very colliery that Professor Moss has been talking about, namely Pendleton. Indeed, I was under cross examination on the subject of that colliery by counsel!

I would like to remark that the conditions at Pendleton are rather peculiar. The down-cast shaft is only eight feet in diameter and at one part it is only seven feet six inches in diameter. There is therefore great difficulty in getting down the quantity of air that is necessary for the perfect ventilation of the workings. I thoroughly agree with what has been said by Mr. Mavrogordato and Mr. Pirow. In the deep part of the mine, where the temperature is highest, I found that the ventilating current was about 14,000 cubic feet per minute. The heat increment, as from the surface to the bottom of the shaft, would be about as follows:

The temperature at the surface was about sixty degrees Fahrenheit when I was there, and it was about seventy-two degrees at the bottom of the shaft, the depth of which is about 1,500 feet or more. At a point in the workings where the level branches away, where the investigation in the workings was carried out, the depth was about 2,400 feet from the surface and the temperature had risen to the neighbourhood of eighty-eight to eighty-nine degrees Fahrenheit. As you progressed further in it increased, so that at the face it was from ninety-two to a hundred degrees.

I have not the least doubt that, if the air current could be increased, the temperature would be reduced to some extent. I agree with the remarks of the previous speaker, that that is the problem we have to face. We have found out, or are in the way of discovering, how to deal with cases of high temperature as affecting the individual, but how much better it would be if we could eliminate the necessity for treating those cases. Difficult as it would be in the case of a colliery where the distance of the workings is so far from the bottom of the shaft and where we have such narrow shafts, yet I think the thing could be met in this way, namely, by the introduction of supplementary ventilation. I speak especially of a colliery well-known to Professor Moss, namely the Florence colliery in North Staffordshire, where by the introduction of a

fan driven by compressed air and stationed some considerable distance in-bye, the ventilation was much improved. That is a way to meet the matter.

Professor Moss has asked a question which I think is deserving of consideration, and it would be most interesting if data could be collected in regard thereto. That is as to the rate of increase in temperature. I don't know of any part of the world, personally, where the heat increment is so great as in the deep coal mines of Great Britain. The heat increment of the deep copper mines of northern Michigan, for instance, is as nothing compared to what it is in the coal mines of Great Britain. High heat increment in coal mines as compared with most (not all) other mines must, I think, be due to the greater rate of oxidation in the former—that is, to absorption of oxygen by carbonaceous matter.

Take Pendleton for instance. I have given you the rates, the various temperatures at different spots. The increase down the shaft is much less than that where the coal is exposed in the workings. I think it must undoubtedly be due to the greater absorption of oxygen.

Some years ago I was investigating temperatures and conditions in the hematite mines of Lancashire, which are comparatively shallow. We passed along a road where the temperature was normal. We put our heads into a hole where was some rock and timber and the temperature there was so very high that we had to withdraw at once. That was due to the decaying timber which was filling the hole.

SIR ALBERT E. KITSON (Gold Coast): I would like to ask Professor Moss if, in connection with this matter, the question of carbon dioxide has been considered. It seems to me that it may have some influence upon the results. This point has been brought forward indirectly by Mr. McEvoy and Sir Richard Redmayne. The purification of air by ventilation removes that trouble. I should like to know if that affects the question, and consequently the results tabulated in the paper. It seems to me, also, that if we were to make comparisons between the results obtained by investigations in different mines we might be able to get much valuable information on the matter.

Then as regards acclimatization, there is no doubt that that is extremely necessary. You cannot expect a man from a hot dry country to come into the dense moist forest and at once do as good work under those conditions of temperature and humidity as he did under the former ones, and *vice versa*. Some natives, we know, are more susceptible than others to the influence of heat, and they are distinctly liable to heat-stroke even in the open. In this respect we know that natives vary, not so much perhaps as Europeans, but they vary to a large extent. I have frequently seen one of my negro carriers almost prostrated by heat-stroke.

Another point brought forward by Professor Moss, an extremely important one, is the adding of sodium chloride to the drinking water to compensate for the excessive loss of salt

by profuse perspiration in the tropics.

PROFESSOR G. H. STANLEY (South Africa): I don't wish to discuss the paper, because I haven't had the opportunity to read it, but there were one or two points raised by Professor Moss and Sir Richard Redmayne which might be dealt with briefly.

The ore in a Rand mine contains in the neighbourhood of three per cent to three and a half per cent of pyrite and the amount of oxidation that it undergoes during mining and transport, etc., can of course be deduced very closely by the amount of lime which it is necessary to add in the mill to correct the acidity. The oxidation of timber cannot be responsible for anything but a very, very small increase of temperature in the air, because the amount of timber used is not very great and it lasts a long while; in some mines, also, the timber is replaced, to a great extent, by concrete and even other materials. I imagine that neither pyrite nor timber would be responsible for a noticeable increase of temperature.

THE SINKING AND EQUIPMENT OF THE VENTILA-TION SHAFT OF THE GOVERNMENT GOLD MINING AREAS MINE*

By A. Job (Member, S. Af. Inst. Eng.) **

(Quebec, Que., Meeting, September 26th, 1927)

In order to improve the ventilation of this mine, it was decided to sink a circular shaft 22 feet in diameter within its walls, and equip it with a fan on the surface capable of inducing a flow of 900,000 cubic feet of air per minute through the mine workings.

The site of the shaft was located at a point approximately midway between the four existing shafts of the mine, and directly above a drive which, at this particular place, had passed through an unpayable area at a depth of 3,405 feet from the surface; and as this drive intersected each of the two main inclines leading from the north to the south vertical shafts, the ventilation would by this means be split into four more or less equal sections, each having a seven-compartment vertical shaft for admitting air into its workings.

Sinking of the circular shaft was commenced during April, 1922, with temporary equipment. When a depth of 25 feet was attained, water began to percolate into the shaft through the bottom and from all around its periphery. As the ground passed through consisted of clay and loose boulders, it was obvious that the walls of the shaft would collapse if pumping was resorted to before they were secured.

Sinking was therefore stopped until the concrete collar of the shaft, together with the lower portion of the air-duct leading from the shaft to the fan site, were completed. When this was done sinking was resumed, and immediately the shaft bottom entered rock, which proved to be dolomite.

^{*}This paper was published in the Journal of the S. Af. Inst. Eng., XXIV No. 5, 1925.

^{**}Resident Engineer, Government Gold Mining Areas, Limited, Transvaal, South Africa.

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From the large cavities which existed in it water poured into the shaft, constantly increasing in quantity as the shaft got deeper, and when a depth of 80 feet was attained it became impossible to lower the water with the three No. 11 Cameron air-pumps then in use.

A single-stage centrifugal pump was then constructed to deal with the water. This pump had no diffuser plates nor any very fine clearances, as it had to handle the unsettled water direct from the bottom during sinking operations. It was attached to and driven by a vertical 140 horse power squirrel cage motor, which was originally constructed to operate an 8-stage Sulzer sinking pump.

The starting panel, with switch and compensator, was placed and operated at the surface. The pump, with its motor, was suspended by a wire rope and tackle in the shaft, and was raised and lowered at blasting time by means of a crab winch on the surface.

The mechanical efficiency of this pump was no doubt very low, especially after it had run a short time, as the impeller only lasted about a week during the time when sinking was taking place. As the wearing parts, however, were easily replaced, the shaft was usually kept free of water. At a depth of 100 feet, the quantity of water coming into the shaft amounted to about one-and-a-half million gallons per day, but at this depth the bottom of the shaft entered a layer of solid rock that was impervious to water, and from there on it was noticeable that the water was abating.

When a depth of 150 feet was reached, the sinking was stopped until the permanent sinking equipment was ready, and while this was being accomplished the sinkers were put to sink a small 10 ft. by 8 ft. rectangular shaft in close proximity to the main shaft, in order that pumping might be carried on from it after the main shaft was concreted. This was desirable because the water was an asset to the mine, and also because it was thought to be unlikely that the concrete walls of the main shaft would be staunch when the country surrounding it again got filled with water.

During the time the small shaft was being sunk, the pump in the main shaft was kept going; this kept the country around it clear of water. When the small shaft reached a depth of 110 feet, an 8 in. by 18 in. triplex pump was installed in it about 10 feet from the bottom. The plungers of this pump were operated by means of a three-throw crankshaft on the surface, and 8-in. square timber rods were used to connect the plungers to the small ends of the crank connecting rods. This arrangement obviated the cutting of a pump chamber in the shaft, and did away with the risk of getting the motor submerged.

During September, 1922, the permanent sinking gear was completed. This consisted of a timber headgear 55 feet high, the shaft collar platform with air-operated doors, the sinking stage, the hoisting engine and the stage hoist, and two Marshall Heine boilers. The sinking hoist had cylinders 16½in. by 33in. with 8ft.-diameter drums geared 2¼ to 1. The stage hoist had the same size cylinders, and four 8ft.-diameter drums, its gear ratio being 16 to 1.

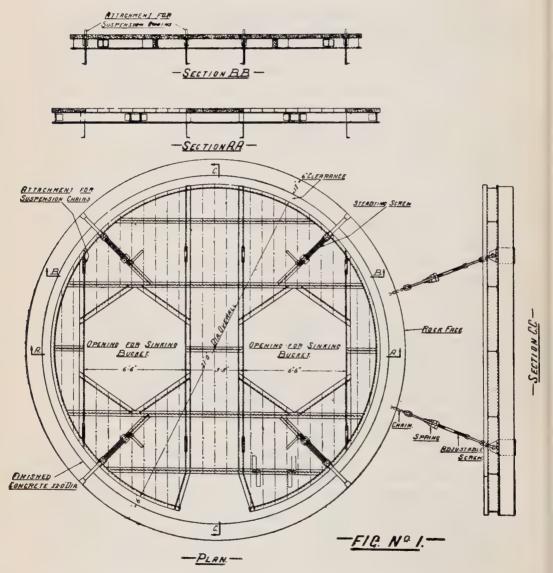
This hoist was reconstructed on the mine out of two hoists that were originally used for sinking the rectangular shafts. The reason for the four drums was that it had been decided to sink the shaft with two buckets running in balance, necessitating four ropes to guide the two crossheads. This was a new departure from the general practice usually adopted in sinking circular shafts, and although disaster was predicted through its use, it proved a success in every way, especially as regards economy of steam, as only one boiler at a time was in use throughout the whole of the sinking.

A further advantage was that, with the four guide ropes, the stage had practically no tendency to turn round, as has often occurred when only two guide ropes are used for suspending the stage; and by having the two sinking ropes and buckets, one could be stopped at any suitable time when the cross-head guide brasses required renewing, or for any other reason, without interfering with the sinking operations.

Two 1½-in. non-spin ropes were used in connection with the three-ton sinking buckets, and four 1-in. diameter, 6/7 construction Langs Lay ropes were used as guide ropes. One set of ropes served during the whole of the sinking, and were in good order when the shaft was completed. All of the drums of the stage hoist were provided with clutches.

This, together with the adjusting screws on the stage end of the rope, permitted the rope tension to be adjusted accurately. Figure 1 shows the sinking stage which was used.

Before sinking was commenced with the permanent gear, the shaft was completely concreted down to within a few feet of the bottom with mass concrete instead of the usual bricking. This was done at a rate of 40 feet per day, excepting for the places where water was coming into the shaft. Here pipes were wedged in the cavities from which the water was issuing, and by this means the water was led out through the space to be concreted.



Holes were then made in the shuttering for the pipes to pass through, and when the concreting was finished and had set, these pipes were plugged with screwed plugs. The concrete was mixed by a mixer on the surface; it was then dumped into concreting buckets which automatically discharged themselves into the sinking stage, or otherwise into the space that was to be concreted.

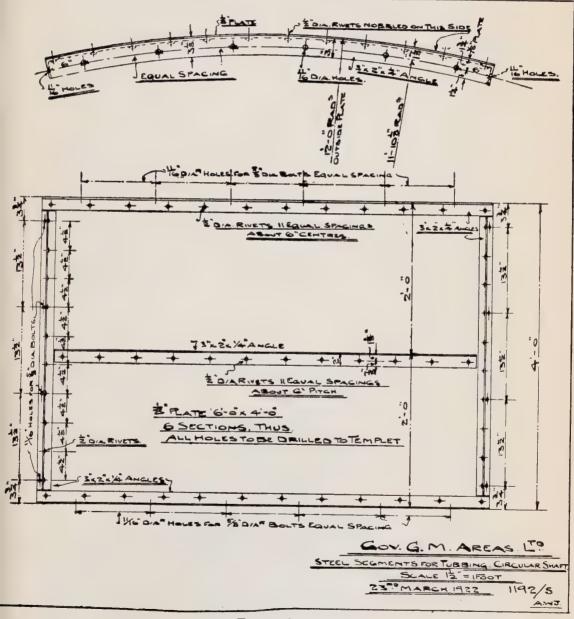


Figure 2

The shuttering used for this purpose was constructed of ½-in. plate four feet deep, and twelve sections formed the complete circle. Each of these sections had angle-bars attached to the four edges for stiffening them and also for attaching them together, which was done by means of taper pines, no bolts being used for this purpose. All of the sections were exact duplicates of one another, and required no sorting when they were being assembled. The design of shuttering used is shown in Figure 2.

When the concreting was completed and all the pipes plugged, it left the shaft practically free of water, which then accumulated in the bottom of the small shaft, whence it was pumped to the surface, and everyone was under the impression that the water troubles were over.

Sinking was then resumed in the circular shaft, and for a short time the rate of sinking was about ten feet per day.

Within a week or two, however, the water again began to find its way into the shaft, and it soon increased to such an extent that two No. 11 Cameron pumps had to be installed on the sinking stage to deal with it. Usually after each blast there was an abnormal inrush of water, due to new fissures being opened up, and occasionally it took days from the time of the blast before anyone could get into the bottom again.

Eventually a depth of 280 feet was attained, and here a pump chamber was cut to accommodate a centrifugal pump which would pump to the surface and so relieve the air pumps of this head of water.

A large ring was formed round the shaft to act as a sump, in which the water from the sinking pumps could be settled to a certain extent. This ring caught a certain amount of water for a time, but eventually the whole of the water came into the shaft from below the ring.

When the shaft reached a depth of 570 feet it became necessary to cut another pump chamber and instal another pump, as the head was again getting too great for the air pumps; but before doing so it was decided to put down a borehole to find out definitely the depth of the dolomite, and if it proved to be only a short distance, effort was to be

made to reach it without cutting another chamber. A diamond drill was procured and a hole was drilled down from the bottom of the shaft a distance of 171 feet before it entered the quartzite; this proved that the dolomite extended to a depth of 741 feet from the surface. Another pump chamber was then cut and a pump installed in it which pumped direct to the surface.

The sinking was then continued until the quartzite was reached, and immediately below the contact a permanent pump chamber and a sump capable of holding a quarter of a million gallons was cut. A ring around the shaft was formed just above this to catch the water that was falling into the shaft, and from this ring it was piped into the sump.

Two 9-stage 'Pulsometer' centrifugal pumps were installed in this chamber, each of which was capable of pumping 30,000 gallons per hour direct to the surface. After these were put into commission, it was found that it required one pump to run constantly, and the second pump occasionally, to handle the water that the shaft was making, which amounted to close on three-quarters of a million gallons daily.

As, up to the present time, this water has only decreased by a small amount, it showed that this quantity at least was what the sinking pumps had to deal with, excepting for a certain amount that was hoisted by means of the sinking buckets; in addition to this the surrounding country was drained down to this depth during the time this portion of the shaft was being sunk. A peculiar thing in connection with this strata is that, up to the present time, the water in the small shaft has never drained itself and, at present, about 200,000 gallons are being pumped from it daily.

The cementation process was not resorted to nor was the thought of it entertained, on account of the water being such a valuable asset to the mine.

After the permanent pumps were put into commission, the air pumps were taken off the stage and sent to the surface. The shaft was then completely concreted down to the bottom, pipes having been placed in the fissures from which water was issuing and then led down through the space to be concreted to the collecting ring above the pump chamber.

The sinking stage was then lowered close to the bottom to be overhauled and repaired, as it had got damaged through rocks striking it during the blasting operations; for, throughout the sinking to this depth, the blasting was done from the stage because the pumps on the stage had to be kept going as long as possible up to the blasting time. The holes were, therefore, charged with long fuses that reached to the stage, and, when all was ready, the pumps were stopped and disconnected from the columns and the fuses were lit and cast off, after which the stage, with the pumps and men on it, was hoisted up to about 50 feet, where the bucket was waiting. The men then got into the bucket and were hoisted to the surface.

During May, 1924, sinking was commenced in the quartzite, the shaft being now practically free from water, as it only made between thirty and forty buckets per day; and with these conditions it was assumed that the shaft would be completed within a year.

Calculations were then made regarding the resistance of the air passages throughout the mine, in view of placing the order for the fan. This entailed a lot of figuring, and a considerable amount of assumption, as the ultimate size and extent of the air passages when the workings extended to the boundaries was unknown. But as far as it was possible to calculate, a depression equivalent to seven inches of water appeared to be the maximum which would be reached. As this would not be required for a number of years, but would gradually be attained, it was decided to drive the fan by means of a steam engine on account of its flexibility as regards speed and power.

Tenders were accordingly invited for a steam-driven fan capable of inducing a flow of 900,000 cubic feet of air per minute through the mine, with a total static water gauge of seven inches.

Numerous tenders were received for different kinds of fans and prime movers, including geared turbine drives, and, after due consideration had been given to the various installations that were offered, it was decided to accept the tender of Messrs. Walker Brothers, of Wigan, for a Walker indestructible fan, which was to be driven by means of a direct-coupled tandem horizontal steam engine. The guaranteed mechanical efficiency of the fan was to be not less than 75.5 per cent under full load conditions, and the guaranteed steam consumption of the engine, when running at full power with 140 lb. gauge steam pressure and 100° superheat, was not to exceed 12½ lb. per horse-power hour.

During the succeeding months good progress was made with the sinking, and in the early part of April, 1925, the shaft holed into the drive at the place intended and within a foot of the calculated depth. The distance sunk in a little less than eleven months was 2,664 feet, which averaged out at over 242 feet per month, the greatest month's sinking being 251 feet.

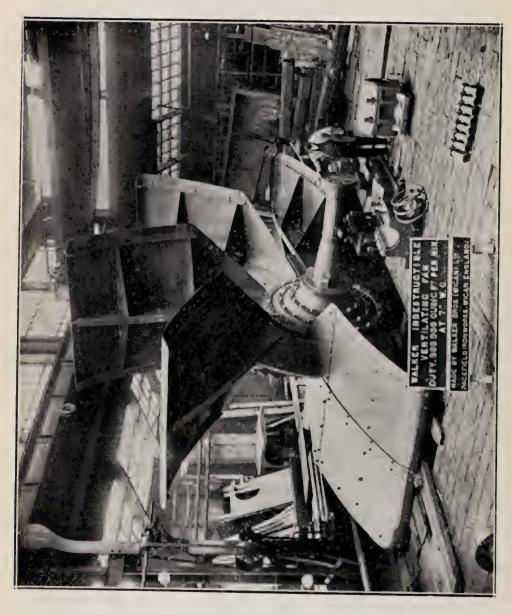
This eleven months' sinking probably created a record, and taking the excavated size of this shaft, together with the quantity of water that had to be baled daily, and comparing it with the size and the relatively dry conditions of other circular shafts on the Rand where greater footages have been sunk in individual months, one must conclude that such progress over a long period was exceedingly satisfactory.

Although it is, perhaps, unnecessary to refer to the work carried out by the Engineering Department in the Head Office of the Group, the writer would like to mention that the detailed arrangements for sinking the shaft, ventilation calculations, and the selection of the fan and engine, together with general supervision of the work throughout were undertaken by the Engineering Staff of the Head Office.

The master sinker in charge of the shaft was Mr. E. Roberts, who is to be congratulated upon the organization and control which made possible the excellent footage attained once the water had been overcome.

Mr. Roberts was, I believe, responsible for the suggestion of balanced winding, which contributed very largely to the results achieved.

After the shaft had holed, the two air passages leading off it were completed down to a certain distance by the sinkers. The shaft was then stripped of its air, water, and ventilation pipes, up to the surface. Before the sinking stage was removed, the deflector at the top of the shaft was fixed, and the air-duct from the shaft to the fan was completed.





Permanent guide ropes were then placed in the shaft, and a small cage was installed for hoisting and lowering persons to the pump chamber. An air lock was erected on top of the shaft in connection with this cage, and the remaining top of the shaft was hermetically sealed so as to prevent any air leakage.

During February, 1925, the fan and engine arrived on the mine, and the erection was proceeded with. The fan impeller is 30ft. in diameter and 10ft. wide. It has eight blades of very robust construction, as shown in Figure 3. The fan shaft is 21in. in diameter in the middle, where the impeller boss is mounted, and from there it is tapered away to the journals which are 13½in. in diameter and 27in. in

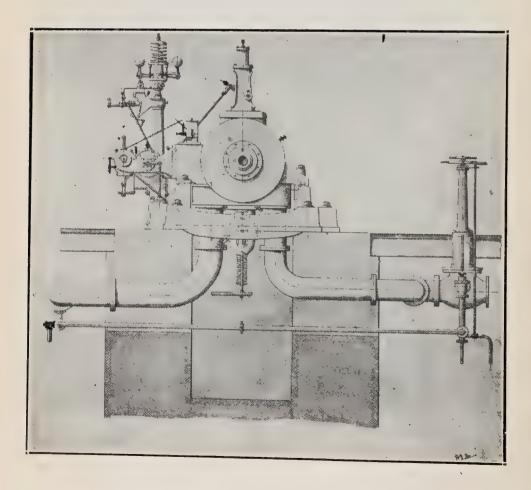


Figure 5

length, the distance between the centres of the bearings being 12 ft. 7 in. The impeller, together with its shaft, weighs over 30 tons.

The whole of the housing was constructed of brick and concrete, except for the évasé roof, which is of steel and is supported between the brick walls. The discharge passage around the periphery of the impeller was formed to an involute curve by unwinding a tape from the circumference of a 6ft. 9in. diameter disc. Its minimum radius was 15ft. 4in. and its maximum radius 36ft. 6in., its width being 10ft. 4in.

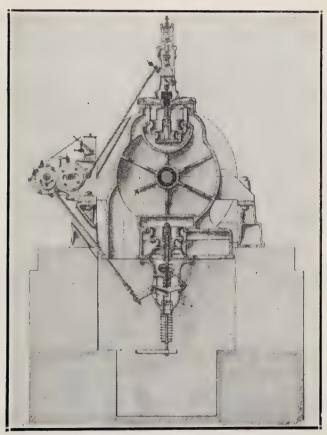


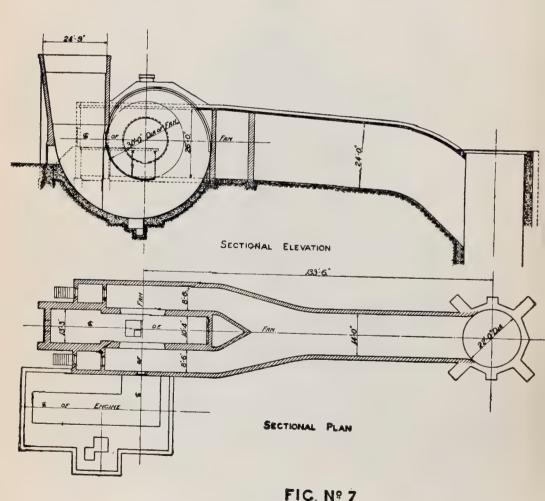
Figure 6

The top of the discharge chimney stands 47 feet above ground level, its internal dimensions at the top being 24ft. 9in. by 16ft. 5 in. The centre of the fan shaft is situated 14 feet above ground level and the lowest portion of the discharge passage is 17 feet below ground level. Each of the two inlet orifices is 16 feet in diameter.

Close on 400,000 bricks were used in building the fan housing and the air-duct leading from the shaft to the fan.

The engine has cylinders 26in. and 52in. in diameter and 36in. stroke, its maximum speed being 125 revolutions per minute; at this speed it is capable of developing 1,500 horse power. Its valves are of the drop valve type, and each valve has four beats. The valves are operated by means of the Doerfel valve gear. The cut-off in the h.p. cylinder is controlled by the speed governor, and that of the l.p. cylinder by hand.

The engine has forced lubrication throughout, and warning is given if the oil pressure falls below 5 lb. per square



inch. The fan shaft and intermediate bearings are lubricated by means of oil rings, and indicating thermometers are fitted to them which give an alarm electrically if the temperature rises above normal.

The engine is very massive in construction and beautifully finished. It is shown in Figures 4, 5, and 6. A Le Blanc jet condenser is used in conjunction with the engine, and this maintains a vacuum of over 22 inches constantly. No cooling arrangement is used with this condenser, as the water from the shaft, which is ample for the purpose, is used for injection, and after passing through the condenser it is pumped direct to the surface plant.

During August of the same year the fan and engine were completed, together with two additional boilers fitted with superheaters. The plant was then put into commission and up to the present time it has run perfectly at a speed of 100 revolutions per minute. This creates a depression equal to $4\frac{1}{4}$ in. of water, which causes the air to flow through the mine workings at a rate of 750,000 cubic feet per minute.

Figure 7 shows the general arrangement of the lay-out of the fan and engine, and the suction duct from the shaft.

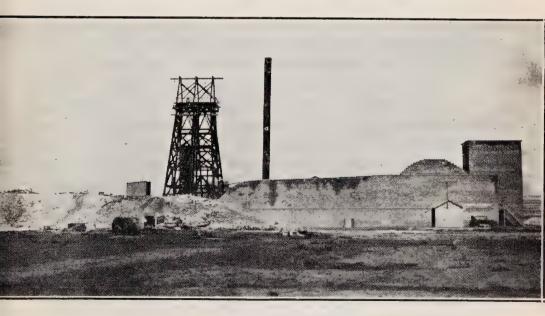


Figure 8

Figure 8 is a photograph of the installation as it appears after completion.

It may be of interest to mention two remarkable accidents that occurred in connection with the sinking of this shaft. The first of these happened when the shaft had attained a depth of about 2,500 feet.

The driver was lowering a bucket out of balance, the hoist having been run the whole distance from the surface with the reversing gear against its direction of motion, its speed being controlled by means of the hand-operated relief valve.

When the bucket was within a certain distance of the crosshead buffers on the sinking stage, the driver closed the relief valve in order to arrest the motion of the hoist. Immediately this was done an explosion occurred, which blew the steam chest of the right-hand cylinder to pieces. some of which went up through the roof, while others went through the floor and also through both sides of the building. The driver's seat, immediately behind him, was split from end to end, a two-inch pipe column supporting a signal bell by his side was knocked over and badly bent, a clutchoperating wheel in front of him was smashed to pieces, and another wheel was damaged. Pieces of cast iron from the steam chest were scattered all around, and the engine room was littered with the asbestos with which the cylinder was The driver escaped practically unscathed. lagged.

The bucket landed on the bottom, and about a couple of turns of slack rope was payed out before the hoist was brought to a standstill by means of the brakes. The driver was under the impression that he applied the brakes after the explosion occurred, but it is my opinion that the disc brakes were applied previous to the steam chest bursting, and this brought the hoist to rest after the bucket had landed.

When I examined the hoist after the occurrence, the drum brakes were applied, and probably the driver applied them after the accident. This took place about 10 p.m., and by 4 o'clock in the morning the hoist was again in operation with a cylinder taken from a similar hoist which was out of commission on the mine.

Figure 9 shows the cylinder after the accident.

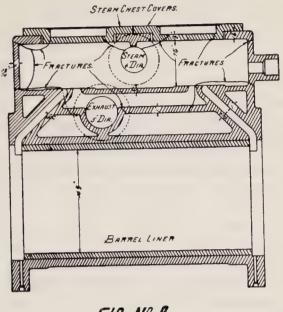


FIG. Nº 9.

Various theories were advanced as to the cause of the steam chest bursting, the general opinion being that it was directly due to the pressure that accumulated in the steam chest after the driver had closed the relief valve. Against this, however, I may mention that the cylinder was fitted with automatic spring load relief valves as supplied by the makers, and these were in working order and set to blow off at about 5 lb. per square inch above the maximum steam pressure.

My contention is that an explosion did take place. In support of this I may state that the sound of it was heard all over the mine at distances of a mile or more from the shaft.

If this steam chest was tested to destruction by any means other than an explosive charge, the weakest part of it would naturally fracture first, and if the fractured portion were large enough to relieve the pressure as fast as it tended to accumulate, then nothing further would happen.

In this case not only did it fracture the weakest part, but the whole of the steam chest was blown to bits, including

portions that formed the exhaust passage and which were never subjected to any other pressure than the exhaust steam on one side and the atmosphere on the other.

As the two steam chests of the engine were connected by means of the steam pipe, whatever pressure existed in them should be common to both, excepting for an instantaneous pressure such as an explosive charge would create; but the other steam chest showed no sign of having been subjected to any excessive pressure.

The divisions between the steam and exhaust ports of the steam chest that burst were undamaged, and this leads one to think that the pressure which burst the chest never reached the steam ports or the cylinder, and that at this particular time the slide valves, of which there were two in each steam chest, were covering the ports and so protected them and the cylinder from damage. Also, I can see no reason why an explosion should not occur in the steam chests of a hoist when it is being run against the reversing gear with a descending load, for it then becomes an air compressor with a defect in it similar to a leaky discharge valve, but which, instead of leaking during the intake stroke, as well as the compression part of the discharge stroke, only leaks during the compression stroke.

When an engine is being run in a reverse direction, its pistons are receding from an open exhaust port and advancing towards an open steam port for a portion of the stroke that depends on the valve setting, and, when the valve opens, the steam port air is admitted back into the cylinder from the steam chest. This air had been compressed and forced into the steam chest during the previous stroke, and as it may then be passed and re-passed from one end of the cylinder to the other and be subjected to excessive wire drawing at certain periods when the valve is opening and closing, its temperature is bound to rise.

The heating effect is accumulative in the same way as in a compressor that has a leaky discharge valve, and therefore it is possible to attain a temperature which is only limited by the conductivity of the cylinder walls. These are usually lagged with some non-conductive material, and as, in this particular case, the oil used for lubrication was forced into the steam chest by a mechanically operated lubricator, all the conditions necessary to cause an explosion were possibly present.

The second accident referred to happened during blasting time.

After 53 holes had been lit up in the bottom of the shaft, the two sinkers got on the bucket and pulled the bell wire to give the signal for them to be hoisted; while the wire was being pulled, however, it broke off a short distance from its lower end.

One of the men immediately got off the bucket to get hold of the other bell-wire which was hanging in the shaft, he being under the impression that no signal had been communicated to the driver by means of the wire that broke.

The driver, however, did receive the signal and hoisted away, leaving this man in the bottom of the shaft. When the bucket was about 1,000 feet from the surface, the driver got a signal on the other bell, which was given by the man in the bottom, but as the driver had no idea that anyone was left behind he brought the bucket to the surface, from where it was immediately signalled to the bottom again. Before it reached the bottom the charges commenced to explode, and the bucket was returned to the surface.

After the explosions in the bottom had ceased, the sinkers went down expecting to find a lifeless body, but were surprised to hear a voice calling; going by the sound they found the man buried by the rock, apparently very little hurt, and quite rational in mind. They brought him to the surface, whence he was immediately taken to the hospital, where it was found that his collar bone was broken; otherwise he was unhurt.

When the shaft was cleaned up after this occurrence it was found that 50 out of the 53 holes charged had exploded.

DISCUSSION

MR. A. S. LILBURNE (Federated Malay States): Mr. Chairman and gentlemen, the paper gives the ventilation of shafts on a very large scale, and this is a matter which is of considerable interest to people who wish to work on a smaller scale. In 1908, when general manager of the South German gold mine in Victoria, Australia, we had a single shaft 2,300 feet in depth and we were confronted with the fact that it had to be ventilated or we could not work it.

The Government suggested that we should put an engine in and drive it with air at the bottom level, and force bad air to the surface, but I decided, with their permission, to instal a suction system. To do this, a twenty-two-inch pipe was passed down the ladder-way compartment, which was amply large enough to allow of it in one corner, to the 600-foot level. There it was taken into the old stopes. We were not working above 1,900 feet and all the levels were closed off with brick work and with pine doors, which could be opened at any time when desired at the various levels and still be kept tight when necessary.

The suction fan on the surface, of 5 feet diameter, was used, and the discharge pipe or suction pipe was carried up 60 feet above the ground level. It was valved to close off when necessary.

In a fortnight's time after this was started, the Government officers came along and tested the whole mine—old stopes, dead levels, and every end that had not been worked for years—for carbonic acid. I am pleased to say that they gave a certificate that the mine was entirely free from it.

That system was continued with success for the two years that I was there and we had no further trouble with the air conditions in the mine. The cost was quite small in comparison with that of the system that was required by the Government and I can recommend it to anybody with a similar type of shaft, where they are able to instal a suction pipe and connect it with their stopes. I think that this is a matter which may be of interest to some of the members dealing with this class of work on a small scale.

PROF. G. A. WATERMEYER (South Africa): Mr. Chairman and gentlemen, this paper is the record of a very magnificent achievement on behalf of the Government Areas and Mr. Job has been very modest in his description of it. I have not actually seen the construction of the Government area shaft but I have seen a similar one on Randfontein, which is being done on exactly the same lines. The contractor has arranged his work so that it is carried on with very considerable expedition. I think the record which has been made there has been 322 feet in one month, thirty days. That, as you see, is a very excellent record. The work is carried out in rhythm. Each individual knows his job exactly and carries it out with expedition.

One of the main features of the platform which Mr. Job mentions is the suspension from four points instead of two by chain. You will notice he mentions the balancing the platform, so as to avoid its getting out of the horizontal. The material is brought down and dumped on the platform. The frames are put in. The concrete is lashed in behind the frames, which are then removed and placed in position for the next set.

We also saw, when we were there, how the last two sections are joined up. That is a very important matter. They are bolted together, but when it comes to the last section a wooden lath is inserted between the bolts so that, after the concrete is set, these sections may be withdrawn first by the removing of this lath, allowing a certain amount of play to prevent the sections from being concreted in.

It is a very great achievement, and Mr. Job is to be congratulated on the way he has presented it to us.

IMPROVEMENTS IN DRILLING EFFICIENCY WITH JACK-HAMMERS†

By P. M. NEWHALL* AND L. PRYCE**

(Quebec, Que., Meeting, September 26th, 1927)

FOREWORD

The investigations which form the basis of this paper are a portion of a series of systematic studies of certain problems connected with underground work, carried out during the past four years on behalf of the mines of the Central Mining—Rand Mines group, and they are used by courtesy of the Directors of that group.

The primary object of the investigations was to find out whether it might be possible to take some measures towards standardization of rock drills and their equipment, thus reducing the number of types and patterns on each mine, and simplifying the question of variety and quantity of spare parts required.

Any rational effort in this direction necessarily involved a determination of the relative merits of the various machines and appliances in use and on the market, more particularly as to drilling efficiencies and dust prevention. The earlier experiments along these lines showed the high relative importance to drilling results of condition of steel, available time, etc., and that there was room for improvement in other things besides the rock drills themselves; and so the scope of the investigations was extended from time to time as indicated by the information gained.

[†]This paper was published in the Journal of the S. Af. Inst. Eng., XXII, No. 8, 1924.

^{*}Assistant Consulting Engineer, Central Mining and Investment Corporation, Limited, Johannesburg, South Africa.

^{**}Rock Drill Research Department, Central Mining and Investment Corporation, Limited, Johannesburg, South Africa.

A preliminary census of the rock drills on the mines showed an astonishing number of kinds in service—as many as ten on a single mine—and new patterns are constantly being brought on the market. Selection of the most suitable is thus a never-ending task, but much progress has been made, and tests are still proceeding.

It is well known that investigations along some line or other are constantly in progress—notwithstanding statements sometimes made by the uninformed that little or nothing is done by the mining industry in the way of trying out new ideas—but we believe the work, with a portion of which this paper deals, is the most comprehensive and systematic study that has been made on these fields in connection with the fundamental operations of mining, that is, the drilling and blasting of holes.

Some indication that work of this nature has been going forward has been given from time to time in the annual reports of various companies, and in the chairmans' addresses, with statements that improvements had been effected; but up to the present little has been said publicly which would give an idea of the details of such improvements or of their possible benefits to the mines generally. It is then with the hope that a brief description of the methods followed, a discussion of the information obtained on a particular subject which has reached a definite stage of development, and a statement of the improvements resulting, may prove of service to others, that this paper is presented.

METHODS OF INVESTIGATION

Before proceeding to deal with the actual subject selected, a few words as to the methods employed will enable the results given herein to be assessed at their proper value.

A small organization was formed on December 2nd, 1919, for the purpose of carrying out the tests. It was headed by a sub-committee (known as the Rock Drill Investigations Sub-Committee) consisting of representatives of the head office engineering departments and mine managers; and special men who could be relied upon to carry out instructions

faithfully were put in charge of detail work, while others, coached to observe all details and record them accurately day after day, were employed continuously as an essential measure in collecting proper data. Observations were taken and logged on all points in connection with the tests, so as to give the actual details as to speed of drilling, air pressures, losses of time and their causes, running time of drills, condition of steel, etc. Throughout the work, the main object has been to ascertain facts—not always an easy matter, especially underground—and to get them in such form as to be unbiassed by preconceived views or personal prejudices, and free, insofar as practically possible, from influence of unknown or uncontrollable factors. It was considered equally essential to secure as much data as the problems would permit of under actual working conditions, so that deductions and suggestions should be on the basis of practical application to routine work. Great care has been taken to ensure uniformity for conditions which could be controlled without vitiating the practical aspect, and to carry on the tests long enough to obtain average conditions for such factors as could not be otherwise controlled or eliminated. In other words, for tests which involved a determination of relative values. all conditions and factors have been kept uniform or parallel. except the one variable under examination. It scarcely needs stating that definite information obtained by these methods provides the only satisfactory basis for conclusions; and conclusions formed upon this basis must naturally have a greater degree of reliability than those based on opinions and statements of parties who may be interested in selling an article, or on deductions made from partial facts obtained by less thorough investigation.

Details of data collected by these methods will be illustrated as the paper proceeds.

THE JACK-HAMMER

The subject of the jack-hammer, and improvements in drilling efficiency with it, has been selected because:

(a) more work has been done in connection with this machine in its varying patterns and methods of use than with any other particular subject;

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- (b) definite results have been obtained;
- (c) these investigations have demonstrated the great possibilities of this type of drill as an economic factor in mining on the Witwatersrand.

The jack-hammer made its first appearance as a stoping factor on the mines of the Witwatersrand about the end of 1916, and until comparatively recently there has not been any pronounced change in the general design, many of the drills in use today being of the same pattern as those first introduced. We should like this to be noted, so that it will be understood that the improvements indicated herein are due only in part to alterations in the design of the machines themselves.

RIGS

Many rigs have been brought out for the purpose of supporting the jack-hammer, and the subject of their design and use was dealt with at length by the Chamber of Mines. The necessity for rigs has never been generally accepted, and a limited series of experiments indicated to us that, under ordinary conditions, the hand-held method gave more satisfactory results. It was, therefore, decided not to use rigs for other tests.

DRY AND WET METHODS

When these investigations were commenced, the ordinary way of operating the jack-hammer was with hollow steel, with air only passing through the steel to the bit, the water for dust-allaying being supplied by a separate hose. Our tests showed very clearly that other methods created less dust and gave higher drilling speeds.

The use of solid steel with a separate water device not only gave lower dust results, but showed a slightly higher drilling efficiency. Hollow steel, with water supplied through the steel, gave still better results, the footage drilled showing an improvement of about 22 per cent over the method with solid steel.

Figure 1 gives the record of daily results obtained with one 40-lb. jack-hammer under the three methods of operating drills above-mentioned. This diagram is submitted to illustrate

-DAILY AVERAGE DRILLING SPEEDS. UPPER CURVE INCHES PER MINUTE OF RUNNING TIME LOWER CURVE. INCHES PER MINUTE OF TOTAL TIME. WET MACHINE. DRY MACHINE,

the method of summarizing the data collected, as well as to indicate the improvement found in drilling efficiency by the 'wet' method. The average speed of drilling for each day on actual running time and on over-all time available for drilling is plotted in inches per minute, the averages for the period, under each set of conditions, being shown by the horizontal lines (running time and over-all time are dealt with later). A notable feature in this diagram is the variation from day to day in the drilling speeds. We are unable to explain this variation, except on the general grounds of irregular quality of bits (heat treatment and gauging, which are treated at length in later paragraphs); imperfect lubrication of the machine, changes in rock hardness, and the personal element of the native operator. This daily variation serves to show how necessary it is to run tests for a considerable period in order to obtain reliable data, and how misleading may be the results of one or two days' work. The data were taken in 1920, and represent conditions at the time of the introduction of the wet method, and before steel treatment and gauging had received the special attention subsequently given thereto. The machine which gave these results was run, for comparative purposes, in parallel with another make of the same weight, which proved to be of a little lower order of merit. The charts for the second machine show irregularities of the same character and degree. The operators were natives, who changed machines weekly, and the machines were daily inter-changed in working position in the stope, so as to make conditions as nearly as possible the same for each. The mean results of the two machines, under each set of conditions were as follows:

TABLE I

				Time		Inches per min.	
Conditions	Holes per shift	Depth of hole	Feet per shift	Drilling	Total avail- able	Drilling time	Total time
'Dry' machines				h.m.	h.m.		
Hollow steel	6.2	*30.3′′	15.7	145	3-32	1.79	.89
Solid steel	7.3	*30.4′′	18.4	2-4	345	1.79	.98
Wet machines.	6.2	41.6′′	21.3	1—53	333	2.27	1.20

^{*}Difficulty was experienced and time lost when greater depths were attempted.

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It will be noted from this table that the total time available for drilling varied slightly under the different sets of conditions. Therefore, the footage drilled per shift is not strictly comparable, but the 'inches per minute of total time' includes the time factor, and these figures, therefore, give a true comparison. As the footage drilled per shift is the figure usually considered, the above figures have been corrected to a common basis of 3 hr. 39 min. total time, as shown in Table II, this time being the average for the whole of the test, which lasted over four months.

TABLE II

Equivalent feet per shift of 3 hr. 39 min. available for drilling	Comparison with dry hollow	
16.2	_	
17.9	Plus 10.5%	
21.9	Plus 35.2%	
	per shift of 3 hr. 39 min. available for drilling 16.2 17.9	

The 'wet' method thus gave a 22 per cent greater drilling efficiency than the solid steel method, even though the bits with the solid steel were somewhat smaller than those with the wet steel. The 'wet' method was introduced on two or three mines of the group with good results, but the solid steel method was at this time preferred on the other mines, chiefly because there was less difficulty in sharpening the solid steel, and the cost of material was lower—half that of hollow.

This series of tests brought prominently to attention the importance of such factors as time available for drilling, heat treatment and gauging of bits, condition of machines, air pressures, etc., the influence of which on the shift's results, or over-all efficiency, was not generally appreciated. These items are of prime importance, and we feel justified in discussing them under separate paragraphs.

'AVAILABLE TIME' AND 'DRILLING TIME'

In Tables I and II the terms 'drilling time' and 'total available time' are used. 'Total time,' as we have reckoned it, represents that portion of the shift between the starting of drilling on the first hole and the completion of the last hole, while 'drilling time' represents the summation of the periods during which the throttle was actually open and the machine running. Certain deductions are, however, made, in arriving at 'total time,' for delays which were not due to operations directly connected with the machine or its equipment, such as stoppages ordered by officials or occasioned by lack of air or water. 'Drilling time' is necessarily much less than 'total time', since a considerable portion of the shift is unavoidably taken up in changing jumpers, moving from bench to bench, lubricating the machine, changing hose connections, and sundry other details, when all is going smoothly; and when jumpers get stuck, or the machine breaks down, or a hose bursts, more time is lost.

The results given in Figure 1 and Tables I and II were obtained from tests that were made during a period when the conditions for a stoping machine to do a day's drilling only gave an available average total time of 3 hr. and 39 min. out of the usual shift of 8 hr. or a little more; this time varied somewhat, of course, with the different methods. This was too small a proportion of the shift to be satisfactory, but, at that period, changes in organization were most difficult to effect. Since the upheaval in the early part of 1922, general conditions have been so altered that, aided by certain amendments to the Mining Regulations made towards the end of 1921, material increases have been brought about in the time available for a machine to operate during an ordinary shift. This, of course, applies to other underground work as well as to stoping with jack-hammers; but the definite figures taken during some recent jack-hammer tests show how much improvement has been effected in this direction, and what this lengthening of the period available means to the work accomplished in a regular shift. These later figures, which will be shown in Table IV, give an average of 5 hr. and 9 min. as the available time, and as they were taken in the same mine, and under similar circumstances,

to those summarized in Table I, they afford a fair comparison. It will be seen that in this later case the running time is still small, being but 2 hr. and 30 min, with the 40-lb, wet jackhammer of the same pattern as that used in 1920. It may seem strange that, with an increase of 1 hr. and 36 min. (from 3 hr. 33 min. to 5 hr. 9 min.) in the time available, the increase in drilling time is only 37 min. (1 hr. 53 min. to 2 hr. 30 min.); but the higher speed of drilling, secured by means of the improvements now to be mentioned, fully accounts for this seeming discrepancy, as the time taken up in changing jumpers and moving from bench to bench is a constant, whereas the time taken to drill a hole of a given depth is less as the speed increases. By measuring the efficiency of a machine in 'inches per minute of total available time' rather than in 'inches per minute of running time,' we get an over-all factor which directly indicates the influence of the time made possible for a machine to operate on the net result of a day's work.

Avoidable time losses, like delays for jumpers, air, or water, are, we venture to think, greater in general practice than is commonly recognized, and a little special study of the subject will probably be well worth the time and trouble necessary to collect and assemble unvarnished facts.

GAUGE OF BITS

At the beginning of the investigations, there was a considerable variation in the gauges of the bits on the various mines, particularly in the case of the jack-hammer bits.

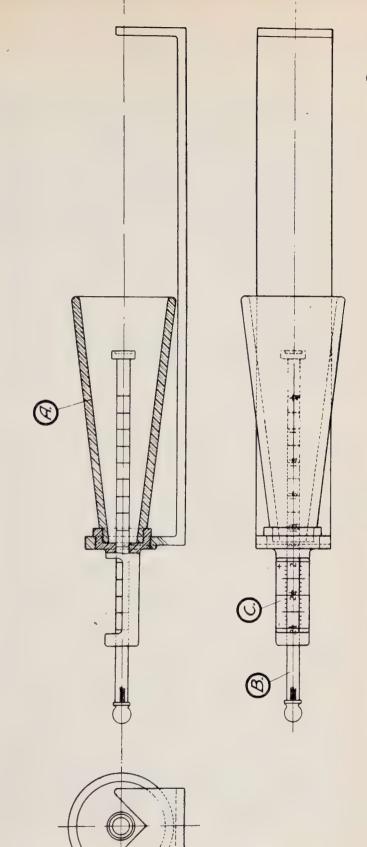
The largest set of bits, which was in use on several mines, consisted of four drills with ¼-in. differences between the gauges of successive bits; while the smallest set, which was used on one mine only, consisted of four drills with 3/16-in. difference between the first three bits and ½-in. between the third and fourth, as follows:

TABLE III

	La	rgest	Smallest					
	Gauge Area		Gauge	Area				
StarterSecondThirdFourth	$2''$ $1\frac{3}{4}''$ $1\frac{1}{2}''$ $1\frac{1}{4}''$	3.14 sq. in. 2.41 " 1.77 " 1.23 "	$1\frac{5}{8}$ " $1\frac{7}{16}$ " $1\frac{1}{4}$ " $1\frac{1}{8}$ "	2.07 sq. in. 1.62 " 1.23 " .99 "				
Mean area	4	2.14 "		1.48 "				

- MEASURING INSTRUMENT FOR ROCK DRILL BITS. -

(PATENT NO. 1142/21.)



F/G. 2.

Taking the volume of a hole to be proportional to the mean area of the bits used, which is sufficiently correct for all practical purposes, it will be seen that a hole of a certain depth drilled with the smallest set of bits will be 31 per cent less in volume than a hole of the same depth drilled with the largest set; and, provided that no trouble is experienced in making the bits follow or in ejecting the sludge, it is reasonable to expect that the smaller hole would be drilled more quickly and with less expenditure of energy.

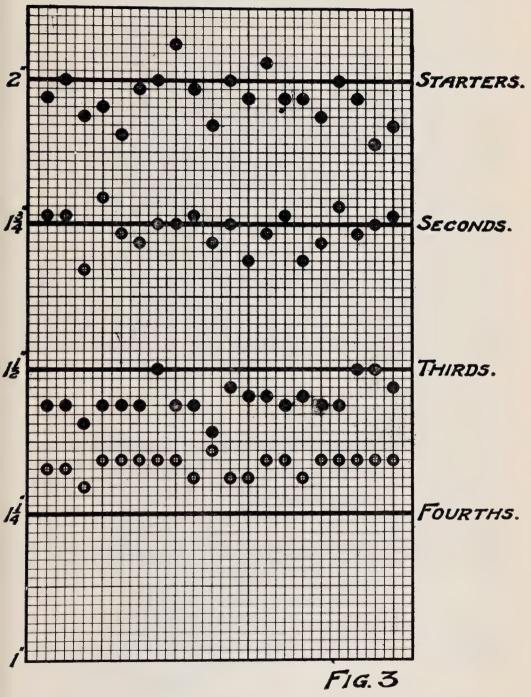
On investigating the matter on various mines, it was found that attempts which had been made to reduce the gauges of the bits had failed on account of the bits not following; and further investigation led to the conclusion that the trouble was due to inaccurate gauging, whereby the nominal differences between the bits were not actually obtained.

To gain definite information regarding the accuracy of the gauging, the measuring instrument shown in Figure 2 was designed. It consists of a hollow cone A, into which the bit to be measured is inserted as far as it will go; the distance the bit enters the cone is a measure of its true gauge, and this is indicated by means of the graduated sliding rod B registering with the scale C.

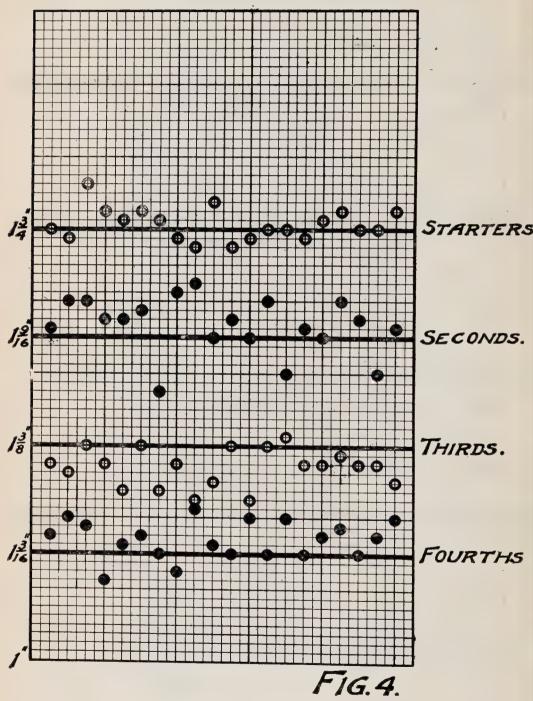
The instrument is so calibrated that the error in gauge, plus or minus, can be read to the nearest 1/64 in. as quickly as the bits can be inserted in the cone.

Samples of the bits on several mines were measured; and to show the effect of the errors in gauge on the differences, the measurements were plotted in chart form, as shown in Figures 3 and 4, which are plotted from actual measurements of bits taken at random. The thick horizontal lines represent the nominal gauges, and the base line the required diameter at the bottom of the hole, namely, 1 in.: the vertical scale being, 1 square = 1/64 in. Each spot represents the measurement of an individual bit plotted in relation to its nominal gauge; for instance, in Figure 3, the first starter measured was minus 2, or 2/64 in. small, the next one was correct, and so on.

GAUGING CHART.



GAUGING CHART.



These charts fully confirmed the conclusion that the accuracy of the gauging was not all that could be desired, and explained why attempts to reduce the differences had failed.

The gauging tool, for use in the sharpening machine, shown in Figure 5, was developed on the City Deep, Ltd., and is now in general use on the mines of the Central Mining-Rand Mines group. It consists of top and bottom swages, bored to a taper of 7° from the centre line, into which stop pieces of various lengths are inserted, according to the gauge of bit required.

If the gauge is to be $1\frac{1}{4}$ in., the bit is sharpened in the ordinary way, but a $1\frac{3}{8}$ -in. dolley is used, and the bit can be from 1/16 in. to $\frac{1}{8}$ in. larger than $1\frac{1}{4}$ in. It is then placed against the $1\frac{1}{4}$ -in. stop in the lower swage and given a blow with the top swage, when the gauge will be reduced to $1\frac{1}{4}$ in. exactly. In addition to being correctly gauged, the bit will then be of the double taper type, the clearance angle behind the points being 7° instead of 14° , as shown in Figure 6.

A chart plotted from measurements of bits treated with the gauging tool is shown in Figure 7. Most of the bits are correct, and a few are 1/64 in. small.

Comparing this chart with that given in Figure 3, it will be noted that the nominal differences have been reduced from $\frac{1}{4}$ in. to $\frac{1}{8}$ in., and that three drills per set are provided instead of four, thereby reducing the volume of the hole by 42 per cent; also that the actual difference between the last two drills is on the average greater than before. It should also be noted that the bits treated in the gauging tool will not be reduced in gauge to the same extent after use, firstly, because they are of the double-taper type, and secondly, because, being smaller, the rate of penetration will be quicker, and they will finish their run in less time.

The gauge sizes shown in Figure 7 have been generally adopted for jack-hammer drills on the mines of the Central Mining—Rand Mines group.

One would imagine that the simple gauging tool shown in Figure 5 would be 'fool-proof,' and that the accuracy of

-GAUGING TOOL FOR BITS.-

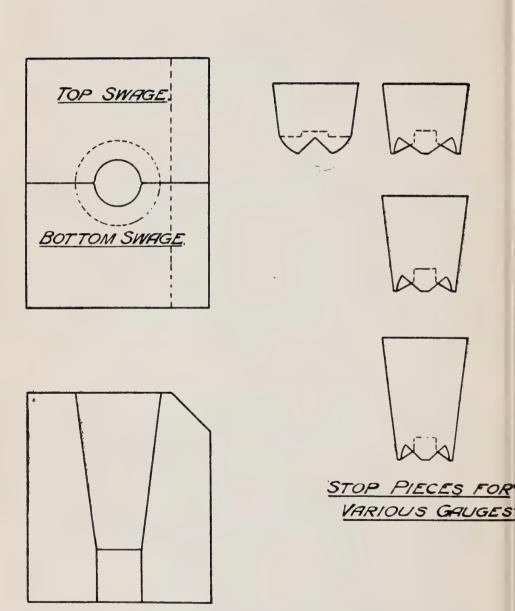


FIG. 5.

the gauging shown in Figure 7 would be obtained consistently without trouble. Unfortunately such is not the case, for many things may happen to cause errors, for instance:

The bit may not be given a sufficiently hard blow between the swages.

It may be smaller than the correct gauge when it is put into the tool.

The swages will wear, causing the bits to be oversize.

The base or cross-head of the machine may be worn, so that the swages are not correctly set.

The bit may not be held hard up against the stop piece, and will be undersize.

Errors arising from these causes have to be detected before the necessary remedy can be applied. It is, therefore,

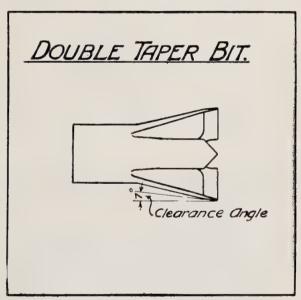
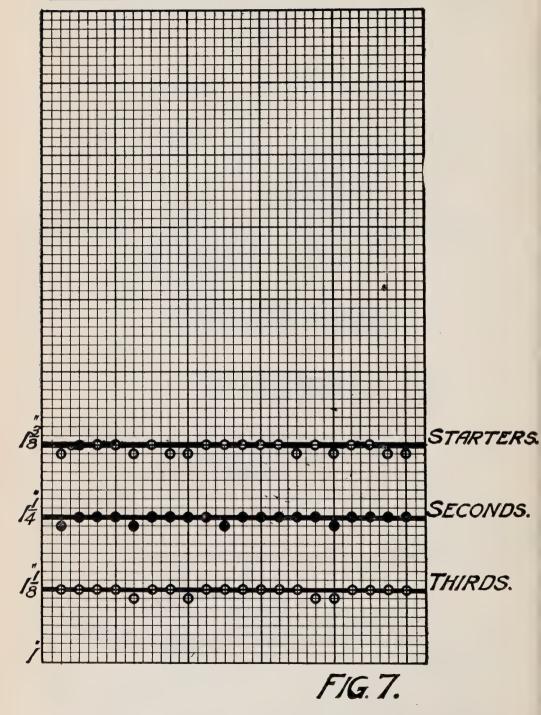


FIG. 6.

imperative, if the required accuracy is to be maintained, that a sample of the bits should be measured and a chart plotted daily.

Twenty sets of drills can be measured and a rough chart plotted on squared paper in about ten minutes by an apprentice, so that the expense and trouble involved is

GAUGING CHART.



negligible compared with the loss of drilling sustained when even a few wrongly gauged bits are sent to the working places. This subject has been dealt with in detail on account

This subject has been dealt with in detail on account of its vital importance, for, as will be seen from Figure 8, the reduction in the volume of the hole, which is made possible by accurate gauging, leads to a considerable increase in drilling efficiency.

The chart shown in Figure 8 was plotted from the daily results obtained with a 40-lb. jack-hammer which was being used as the control machine in a comparative test of several

other patterns.

During the test, the gauges of the bits on the mine, which had already been reduced, were again altered from $1\frac{1}{2}$ in., $1\frac{3}{8}$ in., 1-3/16 in., to the standard $\frac{1}{8}$ in. differences, viz., $1\frac{3}{8}$ in., $1\frac{1}{4}$ in., $1\frac{1}{8}$ in.; the reduction in the volume of the hole being 15 per cent. A comparison of the average drilling speeds before and after the alteration to the gauges shows that the actual drilling speed of the machine on running time was increased by 49 per cent, and on the over-all time by 31 per cent.

Several other tests, covering larger and smaller reductions

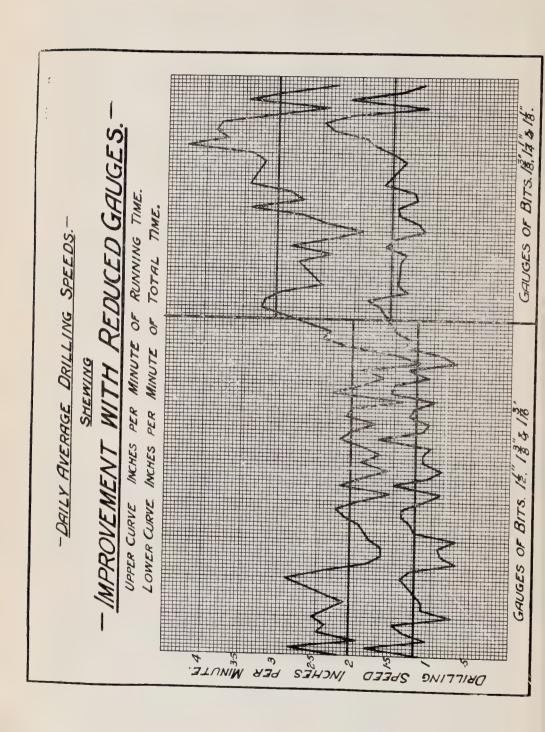
in volume, gave results of the same order.

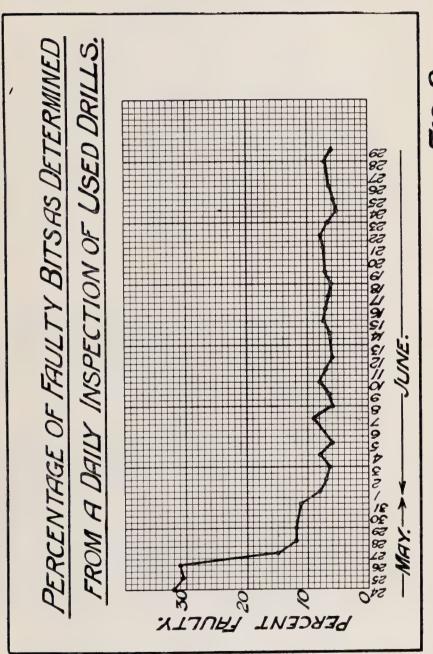
HEAT TREATMENT OF BITS

Under this heading it is not proposed to describe the correct treatment for bits; this has been done many times, and is generally well known. It is only intended to show that, to ensure the heat treatment being correct, it is necessary to supervise this work by means of regular inspection methods, as already mentioned in connection with the gauging.

The appearance of a bit after it has been sharpened and hardened gives no indication of the heat treatment it has received, so that in this case the bits must be inspected after they have been used, when those which are burnt, brittle, or soft can be readily detected, and the percentage faulty ascertained.

During the early stages of the investigations, an examination of the used bits or 'stumps' on several of the mines showed that the proportion faulty varied from 14 per cent to 45 per cent.





F/G. 9.

It was at first thought that this serious state of affairs was due to the use of several grades of steel with different carbon contents, but it was afterwards found that the trouble was generally due to want of adequate supervision, and ignorance of the fact that so many of the bits were faulty. This statement is substantiated by the chart given in Figure 9, which shows the percentage of faulty bits on one of the mines of the Group from the date of starting a daily inspection. It will be seen that for the first three days the proportion faulty was over 30 per cent; on the fourth day some of the first drills to receive more careful attention were returned to the shop, and the proportion fell to 15 per cent, and subsequently to about 7 per cent, at which it was maintained.

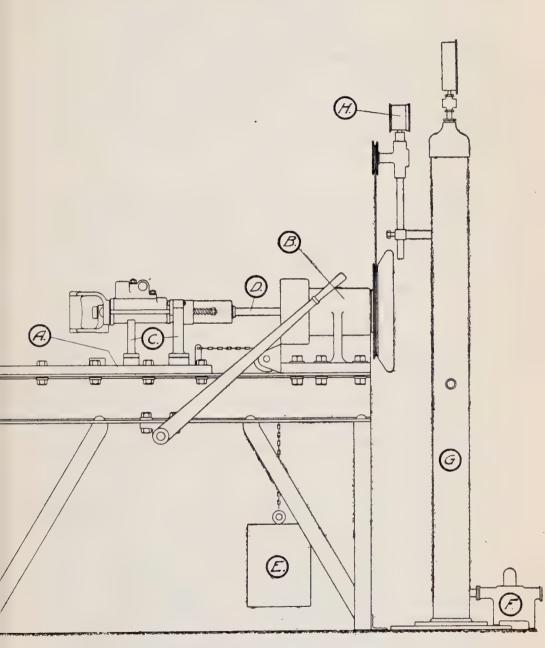
A similar improvement was effected on several mines where a daily inspection was conscientiously carried out.

CONDITIONS OF MACHINES

During the jack-hammer tests, one outstanding difficulty has been the uncertainty which exists regarding the condition of the machines—whether they are in fairly good condition or whether they need over-hauling; and frequently the question is only settled when a machine fails entirely, due to a broken part.

So many variable factors are involved in rock drilling that good or poor results cannot be attributed definitely to the machine. To eliminate this condition of machine factor as far as possible, all the tests have been started with new machines; but, as will be seen from Figure 1, the daily average drilling speed of even a new machine during the first week varied from 1.65 to 3.1 inches per minute, a variation on the minimum of 88 per cent. It should be noted that this variation took place during a test in which every effort was made to maintain constant conditions from day to day.

Consideration of the above leads to the conclusion that, in the absence of some reliable indication of their condition, machines, under ordinary working conditions, will not be sent up for repair until their efficiency has fallen considerably, or until they fail entirely.



-ROCK DRILL TESTING MACHINE. (PROVISIONAL PATENT NO. 738/23.)

A similar difficulty is experienced in the repair shop. It is customary for the drill fitters to test a repaired machine, before sending it underground, by running it against a block of steel for a few seconds. This test is very unreliable, because the machine is not called upon to develop the power necessary to rotate the steel against the friction met in drilling a hole; and it frequently happens that a machine which appears to be satisfactory under this test will not drill at all when received underground. It is, therefore, probable that the efficiency of many of the machines sent out of the shop is appreciably below the maximum.

Under these circumstances, the average efficiency of the machines during their stay underground cannot be very high, and it is evident that some satisfactory means of testing the machines is needed, both before they are sent underground and periodically afterwards, to ascertain when they need further repair.

A testing appliance to measure in absolute units the power developed by the machine, or to compare one pattern of machine with another, is not necessary for this purpose, and would be fit for use only in a laboratory.

It is only necessary to compare, on some arbitrary scale, the performance of the machine under a standard set of conditions to those of drilling, with its own performance, when new, under the same conditions.

The device shown in Figure 10 was designed for testing jack-hammers and Leyner type machines on these lines. (A description of this testing machine is given in Appendix A.)

The machine under test is operated under a standard set of conditions regarding air pressure, frictional resistance against rotation, feed pressure, and constant buffer resilience representing penetration of the rock; and the speed of rotation is indicated on a speedometer. The maximum speedometer reading obtainable with each pattern of machine being known (from trials when new), the state of efficiency of the machine under test can be readily assessed.

DRILLING WITH JACK-HAMMERS—NEWHALL AND PRYCE 663

The speed of rotation under load was taken as a measure of the machine's efficiency for the following reasons:

- (a) It is more easily determined than the blows per minute or the strength of blow.
- (b) If the speed of rotation under load is at the maximum, it is certain that the rotation mechanism is in good order, and that sufficient power is being developed to give the maximum number of full strokes per minute; also, under these conditions, it appears reasonable to expect that the hammer velocity, and consequently the strength of blow, will also be at the maximum.

Several of these testing machines have been installed recently on the mines of the Central Mining—Rand Mines group; and, although it is too early to observe any definite result, there is every indication that, if the machines are tested systematically, their condition, being no longer a matter of guesswork, will be generally improved.

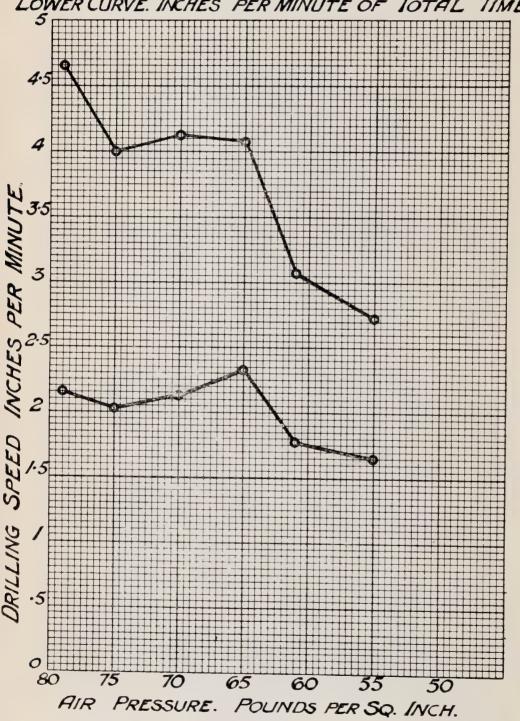
AIR PRESSURES

During a comparative test of two cradle hammer drills, one of which was a standard 18A Leyner, the air pressure was at first maintained at 80 lb. per sq. inch. The test was continued for 41 shifts at this pressure, and on analysing the daily results it was noticed that an unduly large proportion of the time which should have been available for drilling was lost on account of steel breakage, shank troubles, etc. The test was, thereafter, continued for a further 38 shifts with the air pressure reduced to 65 lb. per sq. inch.

A comparison of the average drilling speeds of the machines during the two periods showed that with the lower pressure the actual drilling speed on running time of the Leyner was increased by 4 per cent, while that of the other drill was decreased by 7 per cent; but over the total time, which includes time lost due to steel troubles, the drilling speed of both the machines was increased, by 25 per cent and 20 per cent respectively, when operated at 65 lb. per sq. inch.

Drilling Speed at Various Air Pressures.

UPPER CURVE. INCHES PER MINUTE OF RUNNING TIME LOWER CURVE. INCHES PER MINUTE OF TOTAL TIME.



DRILLING WITH JACK-HAMMERS-NEWHALL AND PRYCE 665

This unexpected result was not considered to be conclusive; but it has since been confirmed by tests carried cut with jack-hammers, summarized results of which are here given graphically.

Figure 11 shows the average drilling speeds of two jack-hammers when operated at various pressures ranging from 55 to 79 lb. per sq. inch.

It will be noted that the drilling speed on running time was greatest at the highest pressure; but figured on the total time a pressure of 65 lb. per sq. inch gave the best results. The various pressures were maintained constant by means of a reducing valve at the top of the stope, and the pressures given were taken at this point; but it was observed that the pressure at the machine when running was from 8 to 10 lb. per sq. inch lower, due to pipe and hose friction. This drop in pressure appears to be excessive, and indicates that the 34 in. hose generally used may be too small for a jack-hammer, a point which is being further investigated.

THE 50-LB. MACHINE

Table IV gives details of a recent test of the 50-lb. wet jack-hammer in comparison with a 40-lb. wet machine of the same pattern as that used in the 1920 tests which are dealt with in Figure 1 and Tables I and II. In this table, the data are summarized as the daily averages over weekly periods and over the whole period.

As an item of interest, we have put into this Table the 'condition of steel' in detail, and this record of run-of-mine steel provides a good illustration of the state of perfection to which drill bits may be brought with careful attention to heat treatment and gauging, the importance of which we have already emphasised.

In 1920, with $1\frac{1}{2}$ -in. starting bit and 1-3/16-in. third, and before heat treatment and gauging of bits had received special attention, the 40-lb. machine gave an efficiency on total time of 1.24 inches per minute. In 1923, with $1\frac{3}{8}$ -in. starter and $1\frac{1}{8}$ -in. third, heat treatment and gauging having been

greatly improved, the efficiency on total time was 2.08 inches per minute. This comparison is not regarded as a strict measure of the value of the improvements effected during the interval in equipment and methods of use of this machine, since the stope in which the drilling took place was not the same, nor have we any means of determining whether the rock was of the same character; but as the tests were carried out in the same mine, and on the same reef in the same portion of the mine, the results may, at any rate, be taken as a good indication of the degree of improvement in efficiency.

Under parallel conditions, the 50-lb. machine shows an over-all efficiency of 2.45 inches per minute, which is 18 per cent better than that of the lighter machine. Both machines were run under practical working conditions, one native operating each throughout the shift, and, as in previous tests, the positions in the stope and the operators were interchanged at regular intervals.

A little objection was, at first, raised by the natives to the greater weight of the 50-lb. machine, but, as they became accustomed to handling it, they expressed preference for it on account of less vibration.

The merits of the heavier type are now quite generally recognised, and the lighter ones are being supplanted by it.

SUMMARY

Comparison between the footage drilled per shift in 1923 with the 50-lb. wet jack-hammer (Table IV) and the footage per shift obtained with the dry jack-hammer and hollow steel in 1920 (Tables I and II) gives a fair measure of the total improvement brought about during the period that these investigations have been in progress. We have endeavoured to show step by step how this great increase from 16 ft. to 63 ft. per ordinary shift, or nearly 300 per cent has been effected, and it should be quite clear that no change in method or machine has done it all.

DRILLING WITH JACK-HAMMERS—NEWHALL AND PRYCE 667

The following table gives concisely the approximate value of each of the various factors in order of progress of development, although 5 and 6 have developed more or less concurrently:

TABLE V

Conditions	Progressive total footage per shift	Per cent value of each improvement
1. 40-lb. dry jack-hammer—hollow steel	16	-
2. 40-lb. dry jack-hammer—solid steel	18	12
3. 40-lb. wet jack-hammer—hollow steel	22	22
4. As in 3—small bits	30	35
5. As in 4—increased available time	42	40
6. As in 5-improved condition of steel and		
machines	53	25
7. 50-lb. jack-hammer, as in 6	63	20

The assessment of percentage value is based on data taken under the direction of the Committee and mainly upon the figures given in this paper, but we have no concrete figures for Item 6. This item is, therefore, given credit for that balance of improvement which could not be otherwise definitely allocated, and the percentage shown is no higher than is fully warranted. The percentages assigned to the several improvements are not suggested as absolute, nor are the footages drilled to be regarded as what necessarily be accomplished by each machine working under the conditions stated; but the footages do represent actual results obtained from an extended series of tests made under working conditions in one section of a mine on the Central Rand, and are, therefore, consistent, so that the value of each improvement as given may be regarded as a fair representation of its merits for general application.

The details of test results submitted herein in the various diagrams and tables form but a small portion of the mass of data collected during the progress of these investigations,

and great care has been exercised to select typical figures—not necessarily the most striking ones—so as to present matters on the most reliable basis possible. It may not be out of place to repeat that the results quoted were obtained under normal working conditions at the various periods, modified only by the presence of observers and by certain requirements essential to keeping conditions constant or parallel.

So far as the 40-lb. machine is concerned: other tests of the series have shown that any one of some half dozen other patterns or makes would have given us similar results under varying conditions, and presumably the same will apply to other 50-lb. machines when available. The figures submitted relate mainly to one pattern of machine so as to provide the necessary elimination of the machine factor as regards method of operating, size of bits, etc., and where the machine has been changed, other factors have been kept constant.

Over-all efficiencies have been expressed for comparison as 'footage per machine shift,' and no mention has been made of the customary term 'fathoms per machine shift,' because other factors, such as the placing of holes and the strength and quantity of explosive used, enter into the fathomage broken. The effect on fathomage of the improvements in methods with which we have been dealing is fairly well illustrated by the results obtained on one of the mines of the group where they have been applied, so far as the 40-lb. machine is concerned. Under the old dry method, the average efficiency over a large fathomage, for the last quarter of 1919, was .52 fathom per machine shift, and for the last quarter of 1923, with the wet method, small carefully-gauged bits, careful heat treatment and longer time available, 1.10 fathoms was the average over a still greater fathomage; and the figure is still increasing, as full benefit is not yet being obtained.

These improved methods are now being generally applied to the mines of the group which use jack-hammers, and the 50-lb. machine is already in use to a considerable extent.

The results quoted for the 40-lb. machine in the preceding paragraph are not isolated, and considerably better results are being obtained with the 50-lb. jack-hammer.

Some experiments have recently been made with the use of two jumpers only in drilling 42-in. holes, and the success attending these efforts indicates further material improvement — possibly not so much in actual drilling efficiency as in the saving in sharpening and handling steel.

Jack-hammers have already largely displaced other machines and hand work in stoping on the mines of the Central Mining—Rand Mines group, except in flat measures in which they are now being tried out. The full scope of usefulness of the jack-hammer has obviously not yet been determined.

The economic benefit to be reaped by increasing the output of work per shift from an individual machine to something approaching the extent shown by the figures we have submitted must be obvious. If an improved method or new type of machine will increase efficiency to such a marked degree, the supporting evidence of detailed costs, to show the actual economic gain, is scarcely required. In any case, costs of a process, such as breaking ground, include so many estimated indirect charges that they cannot be considered as reliable as figures obtainable on drilling efficiency.

Experience has shown that greater difficulty attends the sharpening of hollow steel than of solid, and the cost of material is greater, so that a set of hollow jumpers is more expensive than a set of solid ones.

On the question of machine maintenance, we have no concrete figures at our disposal which would show that, on equal footage drilled, the cost is greater with the wet machine than with the dry. Air consumption, machine maintenance, etc., become items of minor importance compared with the better utilisation of energy, both human and mechanical, obtained under the operating methods described; but, although of minor importance, they should nevertheless have constant careful attention.

It may be thought that the greatly increased footage shown with the latest improvements would require very much greater physical effort on the part of the native operator; but, although this is a matter which does not lend itself to accurate determination, it seems probable that this is not the case. Maximum fatigue is experienced when the machine is running, for, in addition to supporting the weight of the machine, the operator has to withstand the vibration. It is a reasonable assumption—in fact, almost a certainty—that vibration is less intense when more rapid penetration is obtained; and, in any case, the actual running time—and therefore the energy—has been shown to be but little greater with the greatly increased footage drilled.

Great as is the benefit shown to be possible of attainment by the general introduction of the improvements herein discussed, we are convinced that even greater benefit would ultimately be derived by the general application of these methods of investigation through the employment on each mine of special men to study its own individual problems. Such investigations would not only provide the manager with definite facts regarding any method actually in operation and a true estimate of the value of a new method or device, but the data obtained might reveal faults of organisation not otherwise observed. More important still—standards of efficiency would be established in such a way as to be most valuable in determining equitable rates of pay upon the basis of what is reasonably possible under given conditions with facilities provided.

We venture to suggest that work along these lines will greatly assist in prolonging the life of these fields by finding sound means of reducing costs of production, so as to enable more and more of the low-grade ore to be worked.

In conclusion, we wish to record that although we, the authors, have borne a large measure of the responsibility entailed in the investigation work that has been carried out under the direction of the Rock Drill Investigations Sub-Committee of the Central Mining—Rand Mines group, the planning and execution of the tests have been greatly aided by those mine managers who have served on the committee, and have been facilitated in every possible way by the managers and officials of the mines on which the various tests have been carried out.

Much credit is due to the assistants who have been directly in charge of the tests and to the observers who have daily logged everything occurring. Thanks are also due to the authorities of the Government Miners' Training School for making possible the use of the school apprentices as observers and helpers on many of the experiments.

APPENDIX A Description of Rock Drill Testing Machine (See Figure 10)

The testing machine consists of a bed supporting a sliding cradle, A, and a bearing B into which is fitted a flanged bush containing a resilient buffer; the bush is free to rotate and the end thrust is taken on a hardened steel friction ring between the flange of the bush and the face of the bearing. The cradle has three vertical holes bored on its centre line into which are placed supports C which are of various shapes to suit the pattern of machine under test. The short jumper D has a standard shank at one end, the other end being flat and of square section fitting into a square hole in a plate on the face of the rotating buffer. The feed pressure is applied by means of the weight E which is attached to the cradle by a chain passing over a small pulley.

The machine is operated by air at a constant pressure supplied through a reducing valve F into a small receiver G, which serves to damp out the pulsations; the outlet from the receiver is connected to the machine by a hose. The rotation of the machine is transmitted to the buffer through the square ended jumper, and is indicated on the belt-driven speedometer H.

The frictional resistance to rotation is considerable, and is mostly due to the feed pressure on the thrust bearing. which is provided with means for continuous lubrication,

DISCUSSION

LIEUT.-COL. EDGAR PAM (Johannesburg): Before discussing this paper, I would like to draw your attention to the fact that it was written in 1924, and since that date the

investigation into the efficiency of rock drills has been continued.

Mr. Newhall was transferred to Venezuela but Mr. Pryce is still in Johannesburg and Mr. Newhall's place has been taken by another engineer. A very considerable improvement has been made over and above the figures quoted in this paper. As an example, I think Mr. Newhall mentions a fathom and a half per machine shift as having shown a great improvement over the figures before 1922. The figure today which is quite common is about three fathoms per machine shift. true that this improvement is not due entirely to alteration in rock drills; it is due to a great extent to the more careful placing of holes and other factors. The fact remains that whereas ten years ago, or even less, we were all satisfied with an efficiency certainly under one fathom per machine shift, today we consider three to four fathoms quite normal, and in many cases on certain mines they are breaking six and eight fathoms per machine shift.

I mention this particularly because it does indicate the extraordinary improvement which can be made by the scientific study of rock drill work. It is strange, in spite of mining's being, I suppose, the oldest industry in the world, that until recently, as far as I can make out, there was no systematic study of the breaking of rock in any country in the world. That is perhaps rather a broad statement, but I believe it to be true. As mentioned in Mr. Newhall's paper, the Central Mining—Rand Mines group started work on this particular question about 10 years ago.

For those who are not familiar with the work on the Rand, I may say that the mines working under that group are breaking something over 1,000,000 tons of rock a month. From that figure it is clear that they are justified in spending money on experiment and that the results, if used by other mines, must be of tremendous value. Up to recently, although mining engineers normally had very strong and definite opinions on the layout of surface plant, the question of ventilation, shafts, and so on, it was the general practice to allow the miner to decide more or less what rock drill he should use, how he should use it, how he should place the holes, what powder he should put in, and all the other details.

I think that most of us, as students, were taught that when it came to breaking ground the best thing we could do was to apprentice ourselves to a miner and learn from him how to break rock. This idea persisted, and from the little I have seen it still persists to a great extent in this country: it was only after the upheaval in 1922, when we really started fresh in the mining industry in Africa, that we broke away from this idea, and our subsequent efficiencies have definitely proved that we were right in doing so.

Over and above the question of investigation of rock drills, which is one of great importance, Mr. Newhall and Mr. Pryce were responsible for bringing out a method of measuring the burdens which were to be placed on the holes, and this alone has proved an enormous success. It enables a man going into a stope, even if inexperienced, to point a hole and get the maximum burden on that hole after a few days' experience.

I remember Mr. Newhall's coming down to see me underground on one occasion and discussing the placing of holes. To test his idea, we each placed ten holes. We got the miner to put the first ten holes in along the lines we considered best and spent much time putting them in, in order to be certain the holes would be what we considered correctly placed. We then put on what we call a 'hole director', and in not one case in the ten did we find that we had placed those holes to best advantage. This was confirmed by comparison of the amount these ten holes broke and the amount broken by ten holes placed the next day by the hole director. It is now universal practice to insist on every hole being placed by a hole director, and the senior official has to instruct the men how much powder and what sort of powder to put into the hole.

The result has been a reduction in the cost of breaking—I can't quote these figures exactly because I haven't the papers with me, but certainly of more than sixty-six per cent on the original cost.

There is one further point of general interest which comes out of this paper, and that is the principle of investigation on a mine. We came to the obvious conclusion that if this committee of scientific men studying a question like rock

drills could make such an immense improvement, it was probable that they could make a similar improvement along other lines. Not only in the Rand mines, but in every group in the Rand, study departments have been formed consisting usually of a highly qualified technical man with three or four assistants of some technical ability and a fairly large number of juniors whose business it is to take purely time studies of the work which is being investigated. By careful compilation of these figures and comparison of the figures from one mine to another, certain conclusions can be drawn, and although the rock-drill investigation lends itself to more accurate figures and is therefore a better instance to quote, I am quite confident that in questions of winding, air consumption, use of steel, and almost every mining question, improvement is possible which will be reached and is being reached by such investigation.

I am sorry that Mr. Newhall is not here, as there may be several questions which come up which he could reply to better than anybody else. I will willingly do my best in replying to any questions which I can answer, and such questions which I can't answer, if any member is sufficiently interested, I can pass on to the investigation committee in Johannesburg and get them answered personally to any individual by post.

THE CHAIRMAN (MR. THEO. C. DENIS): Both the paper and Colonel Pam's discussion on it have been extremely illuminating. When it comes to breaking down two or three times the amount of rock with the same labour and the same amount of powder, it means that research work can be carried on underground, in shafts and stopes, as well as in the laboratory with microscopes and other apparatus. We are extremely grateful to Colonel Pam for his very clear discussion on the paper, and, in view of the importance of the subject, I hope others will now take up the torch and carry on the discussion.

Mr. G. J. V. CLARENCE (Johannesburg): I haven't very much to add to Colonel Pam's very able introduction to this paper, but there is one point he did not stress which we should certainly put forward very strongly, namely, the urgency for as many Canadian mining men as possible to visit South

Africa in 1930. I am certain that any Canadian mining company which paid all the expenses necessary to send a man to visit the Witwatersrand would be repaid a hundredfold in the benefit it would derive from what he would learn. It is very difficult in a paper or in conversation to make things as clear as when they are actually seen in operation.

The mining man who is particularly interested in certain subjects can, by going down our mines, see in detail the actual operations in which the efficiency has been so greatly improved and realize how, in working, they become merely routine in many cases. His attention being thus drawn to certain points he would see how they could be immediately applied to similar problems in the mines of Canada. Much has been done on the problem of breaking rock efficiently and I am sure that when the Canadians come out they will be amply repaid by what they learn on this point alone, though, of course, there are a great many other points from which they would gain benefit.

SIR ALBERT E. KITSON(Gold Coast): I should like to mention that the use of the jack-hammer drill has now been extended to the Banket mines in the Gold Coast with great advantage to the work. Costs have been reduced very much, and although the native miners are comparatively new to the use of this drill there is no doubt that they will improve quickly and costs of mining be still further reduced.

THE CHAIRMAN: From the Gold Coast, where do we go? Colonel Pam has kindly offered to answer questions that may be put to him, so this is a great opportunity for clearing up any obscure points.

PROFESSOR R. K. WARNER (United States): Inasmuch as the jack-hammer is used in the United States, I would be much interested in knowing what types of drills are mainly used in Africa.

In passing, I may say that, in the United States, jack-hammer is spelt without the hyphen and with only one 'm'. That was the original name.

LIEUT.-COL. PAM: The question of spelling the word jack-hammer is not within my province, and there can be no harm in a difference between the United States and ourselves.

With reference to the question as to the types of machine used, I think nearly every make of jack-hammer which has been made in England, Canada, or the United States, has been tried out on the Witwatersrand. The Ingersoll-Rand, the Waugh, and other machines, have been imported from America and the Holman and Climax are the most important of those from Great Britain.

In addition, certain machines are made in South Africa, which possibly combine the best features from the foreign makes.

I think the greatest number of machines used in Africa are the Ingersoll-Rand and Holman drills.

MR. A. S. LILBURNE (Federated Malay States): I would like to ask Colonel Pam a question. In the development of jack-hammer use, it states clearly in the paper that no rigs of any description are used. They use it with the hand only and apparently quite successfully. That is a method that I don't think is commonly used elsewhere and I would like to know whether the ordinary miner can use the jack-hammer without any difficulty in this way.

Also can they be used outside stoping, for development work, driving, and that class of work?

LIEUT.-COL. PAM: I would like to make it clear that the South African engineers do not claim to have designed or perfected the jack-hammer, but they do claim that they are using it most efficiently, and that they have studied questions of air pressure, steel, etc., to great advantage.

As a general rule, jack-hammers are held without the assistance of a bar or other form of rig, although many of these have been tried out.

Jack-hammers have recently been used on development work and have proved successful from an economic point of view: they have not been as successful as heavier machines in speed, but where the necessity is for economy rather than rapidity, jack-hammers are undoubtedly replacing the rig type in many mines.

MR. H. O. DIXON (England): I would like to ask if the Research Department of South Africa has any figures as to the relative efficiencies between the collared-shank-type steel

and the plain-ended steel. I noticed that many of the mines in Canada are drilling with the anvil blocks and square-ended steels without any collar whatever. I would like to know whether there is some difference between the efficiencies of these two types and whether the collared-shank type is the popular one on the Rand or whether the plain-end is the popular one.

LIEUT.-COL. PAM: Generally speaking, the jumpers used on the Rand are made up with a shank and ring, and the machines are used without anvil blocks.

MR. HENRY WALKER (England): When I was in the United States I was in a colliery where they studied very carefully how holes should be put in to bring the biggest burden. They had a blue-print that was given to each shot firer and he had to see that the holes were put in exactly as that blue-print showed. I think also in the United States they are finding out now how to get rounder coal, bigger coal, by studying the way in which the holes are charged. A space is left between the charge and the stemming.

There is another point in connection with jack-hammers which may be of interest, and that is that in South Wales, in order to avoid shot firing, they are being used to bring down the roof instead of using explosives, and so are an aid to safety.

MR. GEORGE A. DENNY (Rhodesia): We were working in Mexico, under rather similar labour conditions to those described by Mr. Lilburne.

As far back as 1914 we were driving cross-cuts to intersect certain veins in a property at Pachuca, and, time being an important object, we wished to increase the rate of advance. The period was one of the many political revolutionary phases. Big rock drills and spares were quite unprocurable. The only alternative was jack-hammers, machines which had little vogue at that time, and it was not without some doubt that I finally decided to adopt them.

As the machines had been used only for sinking, up to the time I bought them, the question of adopting them for drifting naturally arose. A cradle was designed and ordered but delivery could not be promised for some weeks. The man in charge of operations at the mine suggested that we should attempt to use the machines as drifters without cradles. After some demur I agreed.

The machines were operated by two men; one man held the machine and pressed and fed it, the other threw water into the hole and assisted generally. The rate of advance by big machines in the district generally did not average more

than 90 to 100 feet per month.

In similar rock, with the jack-hammer, we advanced at an average of from 150 to 160 feet per month.

I might add that the cradles were never used. Also that we had used the jack-hammers for sinking in wet ground quite satisfactorily prior to the cross-cutting era.

MR. LILBURNE: First, I want to thank Mr. Denny very much for his remarks. What I was chiefly interested in knowing was whether the jack-hammers were generally hand-held. That appears to be the case.

As far as dust is concerned, as we use hollow steel now-a-days that is eliminated; but I have native labour, and if the native labour of the Rand can use these jack-hammers so efficiently there is no reason why my labour cannot do the same. I only wanted to know whether, instead of using cradles of any description, they are hand-held. As that appears to be the case, that answers my question.

MR. H. S. DENNY (Canada): I should like to say that the point that impressed me most, the most illuminating thing in what Colonel Pam said, is that despite the fact that they had jack-hammers and knew a great deal about them several years ago, the extraordinary improvement in costs had been made, not by improved mechanical design of the drill, but by other things.

He has pointed out what happened in the case of the hole director, and the improvement that was made by the skilled observance of conditions and the study of what they were doing rather than leaving it to the miner. That, to me, is most illuminating. It shows that underground, where we have been satisfied to follow ordinary observance, leaving

our miners to fix the holes and place the amount of powder, there is a wide field for careful study and research that will lead to wonderful economies. It wasn't the jack-hammer. The jack-hammer is a wonderful machine, but it had to be supported by this study, and that, to me, is the illuminating thing.

The possibility of improvements in metallurgical practice has appealed to us, since we can work in the daylight; but when we have had to go underground and study things down there, research has not seemed so attractive. Yet we know the bulk of our expenditure has been underground. This study of the cost underground, and going into these details so carefully, to me is the important point of this paper.

THE CHAIRMAN: Is there any further discussion?

LIEUT.-COL. PAM: I should like to make a remark with reference to Mr. Walker's contribution recommending the use of blue prints.

Experience indicates that the miners find great difficulty in reading and using a print, and for this reason we use a director by which the holes can be placed correctly.

The use of directors for development work is more recent, and not so perfect as that in stoping, but it seems probable that in the near future the direction of all holes, whether stoping or development, in South Africa and in other countries will be controlled by means of a mechanical device: I believe that in the big tunnels in Switzerland a similar, though more elaborate, scheme was used many years ago.

MR. LILBURNE: I think the whole discussion boils down to this: As far as the drilling in the Rand is concerned, they have achieved certain results. Every man running a mine has to study his own particular ground to get the best results and there is no actual rule that can be laid down for that. Every man has to study the breakage of his own ground and when he finds he gets certain results in a given way that are better than any results obtained in any other way, that is the line of working to follow. Any actual system of working can only be carried out on practical working lines, which experience of the particular ore-body shows as the most suitable method.

MR. DIXON: There is one point that occurs to me and that is whether you train young men solely to do this drilling or whether this work is given to all the miners. I have in mind our own mines where there is very hard rock roof and floor. Ten holes six feet deep out of every machine per shift, that is with two specialised men per machine in very hard rock, is the result of this training.

LIEUT.-Col. Pam: Two points have been raised which I would like to reply to. Firstly, the theory that each man gets to know the ground on which he is working and uses his experience successfully in getting better results. Our experience is that this is an exploded superstition, and that the personal error due to lack of scientific knowledge is much greater than the variation in the ground in any district.

I think this conclusion would be found to be true within any given area in any part of the world.

The Department, which has studied the question scientifically, cannot lay down definitely what burden would be most advantageous for every individual hole, but better results are achieved by fixing an average burden than by allowing individual miners to decide for themselves.

The second question raised was as to the South African labour being particularly selected and experienced in the work of breaking rock. Experiment has proved that any young man coming more or less new into a mine after perhaps a year of experience is more successful as a breaker of ground than the old miner.

One does not wish to make unkind remarks about the old miners, but the fact remains that many of them refuse to be taught new ideas.

MR. T. C. FUTERS (England): I wonder if we can be told how far the idea of accurate finishing of the drills was carried out. I notice in the paper that special gauges were made for sharpening and I quite believe that a good deal of efficiency would result from that as compared with the haphazard sharpening of drills of particular size and gauge.

LIEUT.-COL. PAM: It is impossible to assign any proportion of the improvement in breaking to any particular cause, and it would be quite impossible to forecast the improvement

possible by the better sharpening of steel without knowing the quality of the steel supplied to start with.

It is quite certain that the improvement of the steel and sharpening was an important factor, amongst many others, in improving results.

ADDITIONAL DISCUSSION

MR. C. J. GRAY (South Africa): Mr. Chairman, I was unable to be present at the discussion on the paper by Messrs. Newhall and Pryce, being in the Blue train which did not arrive until late.

In conversation with various friends with reference to that discussion, it has been made clear to me that a certain important feature in connection with practice in the use of jack-hammers in South Africa was not made plain to all. At the time the tests were made, it was the practice, in all jack-hammers going under the name of 'wet' jack-hammers, to use exhaust air as well as water through the hollow steel. In present practice every attempt is made to prevent air accompanying the water through the hollow steel. In fact, its use is now prohibited in South Africa because it has been found that when air accompanies the water through the hollow steel large quantities of the exceedingly fine dust which causes miner's phthisis issue from the hole being It has been found that the air can be eliminated without interfering with the drilling speed of the drill. Records in drilling efficiency are now being obtained with water only through the steel which greatly exceed those records obtained when air also was used. I think, therefore, we can credit ourselves in South Africa with eliminating to a great extent the dust danger in drilling, with a gain in efficiency. That may be interesting to the members of the Congress.

A NEW FORM OF AIR METER AND THE MEASUREMENT OF COMPRESSED AIR*

By E. J. LASCHINGER (Member, S. Af. Inst. Eng.) **

(Vancouver, B.C., Meeting, September 15th, 1927)

For the last 20 years or so, the author has been particularly interested in the subject of the measurement of flow of compressed air.

This problem seems to be quite simple, but I am sure that those who have studied the subject, and who have experimented or have had practical experience with air meters, will agree that the accurate measurement of compressed air is difficult—more difficult than measurement of liquids, such as water—and we have all had our experiences with regard to water meters.

Fortunately, scientific experimenters have given us a fairly accurate knowledge of the laws governing the behaviour and physical constants of compressible fluids, upon which we can base fairly accurate deductions and, in the mind's eye, visualise and measure invisible substances such as air,

The measurement of compressed air and the latent power it possesses is a very important matter from the practical and commercial standpoint, in view of the amount of power thus used, especially in mining operations, and its cost, which is high when compared with electric power for example.

On the mines of the Rand there must be about 300,000 h.p. of air-compressing machinery installed and at work,

Owing to its generally harmless nature, air power is more wasted than any other kind and, owing to its invisibility and to absence of noxious or dangerous effects, such as those

^{*} This paper was published in the Journal of the S. Af. Inst. Eng., XXIII,

No. 7, 1925.

**Water and Power Engineer, Central Mining and Investment Corpora
South Africa Past President of the S. Af. Inst. Eng.

of leakage of gas or electricity, it is difficult to discover or trace; or, if wastage is known to exist, it is generally neglected or left for a more convenient season to be rectified.

The various types of meters well known to the author may be briefly indicated as follows:

- 1. The theoretically simple method of filling a vessel of known capacity with air and discharging it.—This method is of value for test purposes but not for commercial use if the air power is required to do work.
- 2. Displacement meters of the piston type.—These generally waste power and give a lot of trouble in practical operation.
- 3. Displacement meters of the oscillating disc type.— These have too much leakage or else stick.
- 4. Measurement by fixed orifices either of the venturi, formed or sharp edged type.—These are not accurate at low flows, because the flow varies as the square root of the pressure difference and they have, therefore, an awkward flow scale.
- 5. Pitot tube measurements.—The same remarks apply as in 4.
- 6. Inferential meters of the vane or turbine type.— These are unreliable due to variation in frictional resistance.
- 7. Measurement by differential orifice such as a disc or other form of obstruction of fixed weight floating vertically in a conical tube.
- 8. A piston of fixed weight moving up a cylinder and progressively uncovering small orifices in the walls of the cylinder.—These give trouble in sticking, due to grit.
 - 9. Meters of the gate type actuated by springs.
- 10. Meters of the weighted door type.—In this type as heretofore constructed, the law of the relationship of gate movement to flow of air is complicated and as generally constructed it is unreliable at or near its maximum capacity, and the percentage of error is greatest at the points where error represents the greatest amount in power passing through.
- 11. Measurement by means of heating the air passing, *i.e.*, by passing electrical current through a rheostat fixed in a pipe and measuring the flow by difference in temperature between the inlet and outlet end. It may be remarked here

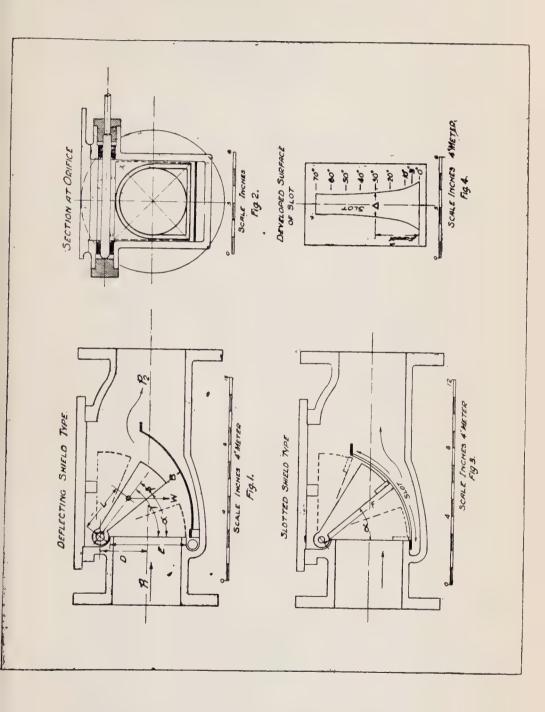
that temperature or heat measurements are amongst the most elusive and difficult things to do with accuracy. It must also be pointed out that the heat (produced electrically) which causes the rise in temperature is entirely wasted, so that the operation of a meter is a source of constant cost. This may seem small, but a little calculation will show that this is not so in the case of large meters. This class of meter can only be used for dry gases, as the least amount of entrained moisture vitiates the results.

The problem presented itself to the author of obtaining a meter that would be suitable, more especially for regular use underground, and after enquiries for meters from many manufacturers and after purchasing a number of different types and testing them, none was found suitable, and the only thing to be done was to design and make one. considering all of the methods of measurement, the author, in conjunction with Messrs. W. H. G. Furnival and D. B. Maclaren of the Rand Mines Air Testing Department, came to the conclusion that an air meter combining the two strong points of the gate and orifice types would be the best for the practical and commercial metering of compressed air-in effect the movement of a gate controlling a differential orifice in such a manner that the delicacy of movement of the gate at low flows should be combined with the relative accuracy of the orifice at high flows. In order to get a positive and reliable action, it was realised that mechanical devices such as springs should be eliminated, and that the weight of the gate itself should be the controlling factor.

Numerous experiments were carried out on different designs, and a meter was evolved, which so far as essential parts is concerned, is very simple.

The theory of the meter, which establishes the relation of the various factors, was to make equal angular movements correspond to equal increments of flow, may now be elaborated.

In Figures 1 and 2 is illustrated what we shall call the shield type, the size of the orifice being dependent upon the distance between the trailing edge of the gate and the curved shield.



THEORY OF F. M. L. FLOW RECORDER

Symbols used for shield plate type. (See Figures 1 and 2).

 α = Angle of swing of gate in degrees.

 β = Angle of lead.

 $\gamma = \alpha + \beta$.

a = Area of orifice, sq, in. = Bb.

b = Uniform width of gate in inches.

c = Co-efficient of discharge.

g = Acceleration of gravity, ft. sec. sec. = 32.1.

h = Difference of pressure, front and back of gate, lb. per sq. in $= p_1 - p_2$.

 $p_1 = Pressure$ in front of gate, lb. sq. in.

p₂ = Pressure behind gate, lb. sq. in.

A = Area of gate exposed to differential pressure, sq. in.

 $B = Span ext{ of orifice, inches.}$

D = Distance, centre of pivot to middle of area A, in inches.

E = Depth of gate exposed to differential pressure, inches.

L = Distance centre of pivot to centre of gravity of gate.

Q = Air flowing, lb. per sec.

R = Gas constant = 53.46 average for Rand conditions.

T = Temperature (absolute) = 461 + ° F.

W = Weight of gate, lb.

From the theory of flow of air through an orifice we have:

$$Q = a c \sqrt{\frac{2 g}{R}} \sqrt{\frac{p_1}{T}} \sqrt{h}$$
 (1)

from the theory of balance of couples:

$$hAD = WL\sin\gamma \tag{2}$$

therefore:

$$\sqrt{h} = \sqrt{\frac{W L}{A D}} \sqrt{\sin \gamma}$$
 (3)

combining (1) and (3):

$$Q = a c \qquad \sqrt{\frac{2 g W L p_1}{R A D T}} \sqrt{\sin \gamma}$$
 (4)

Let
$$K = \sqrt{\frac{2 g W L p_1}{R A D T}}$$
 (5)

Then
$$a = \frac{Q}{c K \sqrt{\sin \gamma}}$$
 (6)

Now, from the condition that the angle of movement of the gate shall be proportionate to the rate of flow,

$$Q = J \alpha \tag{7}$$

in which J is a constant determined by allocating to a certain maximum flow, a maximum movement, *i.e.*,

$$J = \frac{Q \max}{\alpha \max}$$
 (8)

Combining equations (6) and (7),

$$a = \frac{J \alpha}{c K \sqrt{\sin \gamma}}$$
 (9)

Therefore B, the span of the orifice for any particular angle of movement of the gate, is

$$B = \frac{J \alpha}{b c K \sqrt{\sin \gamma}}$$
 (10)

From this equation (10), the dimensions of the opening are determined, as all the factors on the right-hand side of the equation are known for any particular case, except the coefficient, c, which is fairly constant and may be assumed at, say, c = 0.8. It can, however, be determined exactly only by experiment. It is known to increase slightly with an increase in the area 'a.'

In Figures 1 and 2 is represented a scale drawing of a 4-inch meter as constructed for a maximum flow of 100 lb. air per min. (1600 cf.) at an initial pressure of 90 lb. sq. in. gauge for Rand conditions—mean temperature 70° F. and maximum movement of the gate of 70° of arc. The gate weighs 8.34 lb. and the angle of lead (β) is 13° maximum drop of pressure across orifice 0.387 lb. sq. in. or 10.7 inches of water and a differential pressure 2.43 inches of water to open the gate at minimum flow.

The other form of gate and differential orifice is illustrated in Figures 3 and 4, which we call the slotted plate type. The plate is circular in shape, conforming to the movement of the trailing edge of the gate, the air passing through a formed slot.

The theory governing the size of the slot is given as follows:

Symbols used in theory of slotted plate gate meter—these are the same as in the former case, except that we introduce the symbols

F = Distance from centre of pivot to edge of gate, inches.

 Δ = Width of slot at any particular angle of opening.

In this case, however, the measurement of angles (so far as the mathematics are concerned) is in terms of arc to radius unity.

We have again:

$$a = \frac{Q}{c K \sqrt{\sin \gamma}}$$
 (6)

and also
$$Q = J_1 \alpha$$
 (11)

therefore:

$$a = \frac{J_1 \alpha}{c K \sqrt{\sin \gamma}}$$
 (12)

differentiating, we have:

$$da = \frac{J_1}{c K} d \frac{\alpha}{\sqrt{\sin \gamma}}$$
 (13)

from the forms

$$d\frac{x}{y} = \frac{ydx - xdy}{y^2}$$
 (14)

and

$$d\sqrt{y} = \frac{dy}{2\sqrt{y}} \tag{15}$$

we obtain

$$da = \frac{J_1}{2 \text{ cK } \sqrt{\sin \gamma}} (2 d\alpha - \alpha \cot \gamma d\gamma)$$
 (16)

since $\gamma = (\alpha + \beta)$ and β is constant

$$d\gamma = d\alpha \tag{17}$$

From Figure 4 we see that

$$a = \int \Delta d F \alpha$$
 so that $da = \Delta F d\alpha$ (18)

Combining equations (16), (17) and (18) we obtain

$$\Delta = \frac{J_1}{2 \text{ c FK } \sqrt{\sin \gamma}} \left\{ 2 - \alpha \cot \gamma \right\}$$
 (19)

converting, so that α is expressed in degrees instead of radians for convenience in using ordinary trigonometrical tables, we have

$$\Delta = \frac{J_1}{2 \text{ c FK } \sqrt{\sin \gamma}} \left\{ 2 - \frac{\pi}{180} \alpha \cot \gamma \right\}$$
 (20)

and
$$J_1 = \frac{180}{\pi} \frac{Q \max}{a \max}$$
 (21)

and as before

$$K = \sqrt{\frac{2g WL p_1}{RADT}}$$
 (5)

Figure 4 gives the theoretical shape of a slot in a 4-inch meter for the same conditions as the shield plate meter, Figure 1.

The coefficient of discharge in this case is about 0.63, but also varies slightly.

It will be seen that when a=0

$$\Delta = \frac{J_1}{cFK\sqrt{\sin\beta}}$$
 (22)

and when

$$\Delta = \frac{J_1}{c F K} \tag{23}$$

As before mentioned, the meters designed on the above principles were intended primarily as distribution meters for the mines of the Central Mining—Rand Mines group of the Witwatersrand goldfields, in order to assess distribution of air to various services or sections of the mines under that administration.

The desiderata in the type of meter required are:

- (1) Reasonable accuracy over the whole range of measurement.
- (2) Robust design to withstand rough handling and the air concussions due to blasting underground;
- (3) Simplicity and few parts;
- (4) As well as being an indicator of momentary flow it must give a graphic flow time record so that a picture of the whole day's happenings is available for examination and for comparison with the operations or work done by the air in the section measured;

- (5) Reliability in continuous operations so that it is not easily choked by air carrying water, oil and a certain amount of dust, etc.;
- (6) It must not have perishable parts such as rubber or leather or depend for its operation on floats in mercury, oil or water. Devices such as springs are also to be avoided;

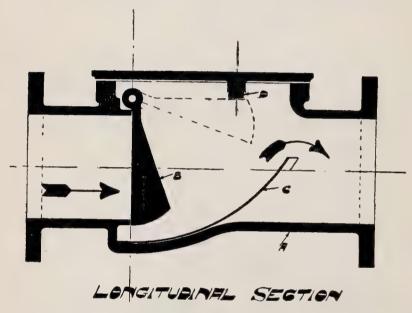


Figure 5.

- (7) It must not be sluggish and when no air is passing should automatically and instantly indicate zero;
- (8) It should require little attention and have a long life and give a minimum of trouble; in short it must be dependable.

After much experimental work, testing and experience gained by practical use underground, the type preferred is of design shown in Figure 1.

The following is a commercial description of the meter called the F.M.L. graphic recorder.

Figure 5 gives a longitudinal sectional view of the essential meter parts in one form of construction.

In the body of the main casting A, a door or gate B, of substantial weight hangs vertically over the pipe opening. Below the lower horizontal lip of the gate is a brass shield C, fairly close to the lip when the gate is shut, but progressively further away as the gate opens further owing to increasing flow of air. The result of this arrangement is to make the angular movement of the gate correspond to the rate of flow so that equal increments of angular movement of the gate represent exactly equal movements of flow over the whole range.

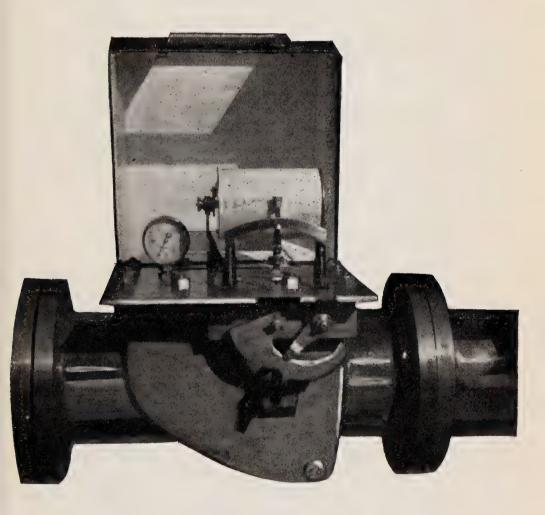


Figure 6.

A pointer attached to the outer end of the spindle of the gate would therefore indicate the rate of flow marked in equal divisions on a graduated arc.

In the meter as built, however, the pointer is not rigidly connected to the gate spindle but the movement is transmitted by an arm to the cam which moves on a separate spindle. See Figure 6. On the top of the cam is fitted an arm carrying a pointer and a pen. The pointer indicates rate of flow on an equally divided scale and the pen marks a continuous record on the chart, which is wrapped round a clock drum in the usual way. Attached to the bottom and side of the cam is a dampening vane working in oil contained in a strong box (not shown). This box also protects the outside moving mechanism from injury and from being tampered with.

The chart is easily changed without risk of damage to the pointer or pen by removing the clock by unscrewing one thumb nut, taking off the used chart, putting on a new one and replacing the clock. The size of the chart has been standardised to 12 inches long to 5 inches wide so as to give a reasonably sized diagram for inspection.

The clock, pointer, pen and indicating scale are covered by a hinged casing with a glass front. This box can be locked up, the meter attendant and some responsible official only having the keys. The glass front is there so that the miner, shift boss, or responsible official can at any time see how much air is being taken, and the record for a few hours previously, and also note the reading of the pressure gauge. A small electric light is provided inside the casing. A spherical level is also fixed to the top plate of the meter to check the true level setting of the instrument.

The meters are calibrated for a standard pressure and temperature, which are the normal for the conditions obtaining. A correcting chart is provided to convert any other pressure or temperature to the normal basis. These corrections are generally small. Automatic devices to correct these introduce too many complications, having in view the object of the meter.

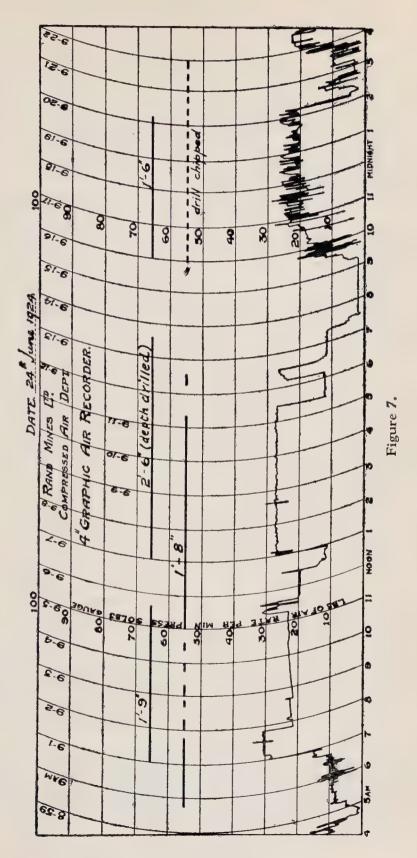


Figure 6 is a photograph of a 6-inch meter with the hinged casing raised.

The meter indicates to within plus or minus one per cent over a range of full capacity to one-twentieth of full load.

The meters are designed for a maximum drop of one lb. per square inch at full load.

All parts are of gun metal or brass to minimise corrosion due either to air or water. The main body of the casting is tested to 200 lb. square inch pressure to find any defects.

Figure 7 is the reproduction of a chart showing a portion of a record taken on a mine on a working shift. The clock was fixed so that the usual one-hour interval represents only 1 minute. Two drilling machines were at work. The heavy horizontal lines represent the time of actual drilling of each machine. It will be noticed that there was trouble with one of the drill bits from 9-16 to 9-22 a.m., and this is reflected in the graphic record of the recorder, whereas from 9-07 to 9-11 a.m., when both machines were working properly, the record indicates by a fairly smooth line. On the Rand Mines, air is measured by lb. air instead of cubic feet.

The outstanding advantage of the uniform scale of flow is that a planimeter may be run over the record, and the total and/or the average rate of air consumption over any period can be obtained. With distorted scales which are common to most recording meters, this cannot be done.

On one mine which has been fully equipped with these recorders on all levels and sections (altogether about 30 meters of various sizes), the sum total of the readings of these meters over a month has agreed from two per cent to six per cent less than the air sent underground as measured in bulk at the surface. The capacity of the compressing plant in this case is 40,000 cubic feet air per minute.

It is therefore, fairly obvious that these meters conform to the desiderata enumerated previously.

It is also to be noted that the path of the air through the meter is direct, *i.e.*, it does not cause the air to turn round sharp corners as in the case of many meters. One effect of this is very little loss of pressure through friction, and the other important practical advantage of not being easily fouled or choked. Dirt and small stones, etc., are simply blown through.

As to reliability and dependability, it may be mentioned that several of the first experimental recorders have been in constant use underground for two years, and when brought back to the test plant for examination and test, were found not to have altered, but were simply cleaned up and sent back again.

There are at present about 300 of these meters in use in these fields.

One point to be remembered, however, is that, in the Central Mining—Rand Mines groups, most of the mines use air from the Victoria Falls and Transvaal Power Co., which is generated by turbo compressors, and is, therefore, non-pulsating flow.

The use of these meters in places where there are violent pulsations is not to be recommended, unless means are introduced for dampening down excessive variations.

This question of measuring air with pulsating flow is most difficult. It has been investigated both theoretically and experimentally, and quite a number of technical and scientific papers have been written within the last few years, but so far as the author is aware nothing final has been evolved. The practical difficulties have not been overcome, although various devices are, and can be, used to minimize the effects.

It is known to those who have studied the problem, that there are two kinds of waves, which we may call pressure waves and velocity waves; the former travel with the velocity of sound and are governed by the same law, but the velocity waves, which form a real pulsating flow, present the greatest problem and cause very queer phenomena, especially in manometer measurements.

It is known that these effects can be minimized by introducing capacities and constrictions in the air transmission mains by receivers, enlargements, orifices and other devices which will absorb the kinetic energy of these waves.

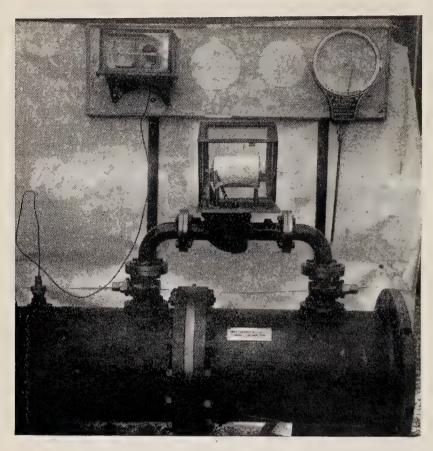


Figure 8.

The author and his colleagues are at present experimenting on certain lines in order to discover some cheap and effective method for dampening down, if not eliminating, pulsating effects, so that some device can be attached to any kind of air meter in order that reliable registration may be obtained in all cases.

A meter operating on the shunt principle has also been evolved, and this is being used with success on three mines on the reef for measuring main outputs of compressor stations.

Figure 8 is a photograph of such a meter. An orifice is inserted in a large pipe and a small sensitive recorder on the gate principle is inserted in a small pipe connecting the two sides of the orifice.

By establishing the relations between the main flow and the shunted flow (similar to the principle upon which most electricity meters are built), the small meter diagram is graduated to read the total flow.

This meter is suitable as a large station meter, and the cost is very moderate. It also has the advantage, as proved by actual trial, of automatically dampening down the pulsation effects of the main flow as affecting the shunt flow, *i.e.*, the pulsations on the recording mechanism are very small compared with the pulsations in the main pipe.

In conclusion, the author would remark that the measurement of compressed air to the various services and sections of a mine is a most important item in checking waste, and in all cases economies are effected. A good many mine managers and engineers have had quite a shock when the result of air measurements were brought to their notice for the first time.

It is, however, necessary to keep continuous records, not merely taking readings at spasmodic intervals, and let the dog get into his bad old ways again through neglect. True economy is obtained by organizing the work of air measurements and supervision, in other words, have the policeman on duty every day.

My thanks are due to the Central Mining—Rand Mines administration for all the facilities given and support accorded in the large amount of experimental work done in connection with air meters and air measurement.

Author's note:—Since the above paper was read before the S. A. Institution of Engineers in January, 1925, water meters constructed on the same principle have been designed, constructed and put into operation, and have so far given every satisfaction. These have also been equipped with integrating gear in addition to the chart record, and are particularly useful as station output meters. The charts show the slightest variation in pump output due to variations in periodicity of electric power supply. The largest size constructed is for a 18-inch main with a maximum capacity of 3,000 gallons per minute with a pressure drop of one pound per square inch. This meter measures with accuracy a flow down to 40 gallons per minute.

Recently, also, the difficulty of measuring pulsating flow has been overcome by means of extending the spindle of the gate on the end opposite to that which operates the recording mechanism. To this is fixed an arm operating a piston in a cylinder and checking the vibrations of the gate by means of an oil cataract. This mechanism is entirely enclosed in a strong box, forming an integral part of the body casting. This box can be kept charged with oil through a plug hole.

DISCUSSION

LIEUT. COL. EDGAR PAM (South Africa): I would like to say a few words on this paper from the point of view of a user of the air meters described. An investigation years ago in Johannesburg showed that the rock drills, pumps, and other air-driven machinery did not account for more than 30 per cent of the total air consumption. Considering the high cost of compressed air, this was clearly serious, and a great effort was made, with the help of meters, to economize. I cannot quote the percentage accounted for on an average mine today, but there is no doubt that the improvement has been very great. The chief uses of the meters are:

- 1. To isolate the consumption in sections of a mine and soplace the responsibility on individual officials and compare the consumption in various districts.
- 2. To test the consumption of various air-driven machines, to see that the machines are maintained in good condition, and, if found more economical, to replace air-power by electrical power in certain cases.
- 3. To check the closing of valves and leakage of pipes.

METHODS OF ELIMINATING BARREN ROCK FROM ORE AT THE SUB NIGEL MINE, TRANSVAAL

By W. A. Quince (Member, Chem., Min., & Met. Soc. of S. Af.)

(Winnipeg, Man., Meeting, September 3rd, 1927.)

The ore-body at the Sub Nigel mine is a narrow one, the pay portions being in well defined shoots. The average 'reef' width is about 12 inches, with a range from 'contact' up to 45 inches; the average value is 40 dwt. per ton, giving an average of 480-inch-dwt. The average dip is about 12 degrees.

There are four methods employed for eliminating the 'waste' rock:

- (a) Resuing development.
- (b) Stoping at narrow widths.
- (c) Resuing stoping.
- (d) Close sorting-out of waste rock on surface.

A.—RESUING DEVELOPMENT

The average cross-section of the drives, raises, and winzes is nine feet wide by seven feet high.

With a 12-inch reef at 40 dwt., the value of the ore from development is roughly about six dwt. per ton when the whole 'round' is blasted.

With resuing, the 'round' is taken out under the reef at a lesser height and the waste trammed away. The reef is blasted down on the next shift, and for a 12-inch reef is sent to the crushers as ore at about twenty dwt. per ton. The standard practice is for each developer to have four 'ends', so that two rounds per shift are obtained. The machines drill over the whole face, and that portion of the

round below the reef is blasted alone, the holes for bringing down the reef being drilled with the round and blasted separately.

This method eliminates about 80 per cent of the waste rock from the pay development, which greatly increases the value of the ore and also brings in most of the low-grade development reef as pay ore. About twelve inches of waste rock is 'carried' with the reef body, six inches above and six inches below; this is necessary to ensure that all the reef is obtained.

The ore obtained by resuing is not only of very much higher grade, but it is broken into sizes suitable for the tube mills. The only disadvantage of the method is that the rate of advanced is halved.

Costs:

There is a slight saving in explosives in the resuing holes, which is balanced by the two trips for cleaning out.

B.—STOPING AT NARROW WIDTHS

In the process of reducing the stoping widths at this mine, the chief difficulty which presented itself was the removal of the broken ore. A fair measure of success has been achieved in the solution of this problem.

Stoping is started from the original raise and winze, up and down which a one-ton skip operates, either tipping direct into one-ton trucks on the drive, or into foot-wall cross-cut boxes. The ore mined is shovelled directly into the skip until the stope faces have advanced about 20 feet from the original raise and winze. The faces are then stopped and stope tracks are cut in the footwall 25 feet apart, and equipped with 16-lb. rail 18-inch gauge track and one-fifthton stope-trucks. These tracks are kept extended right up to the working face. The waste shot up in cutting the stope tracks is packed above and below the tracks, all fines being filled into the packs. During blasting operations, the stope trucks are left in the original raise and winze, out of the way of the blast.



Figure 1.—Machine boy drilling in a stope with an average stoping width of 16 inches. By the 24-inch rule, it will be seen that the stoping width near the face is only 12 inches.

In cutting the stope tracks, sufficient footwall is shot up to give 42 inches from the hanging of the stope to the top of the rail. Two holes for the tracks are drilled from the track end towards the face as lifters, being 3 feet apart and from 4 feet to 5 feet in length, and charged with one 40 per cent primer each. This breaks the ground suitably for packing, very little fines being produced.

The track holes are drilled by the stoping boys as they pass, and are drilled ahead of the track under the faces, so that, as soon as the face has advanced sufficiently, the track holes are blasted and the track again extended to the face, 6-foot lengths of temporary track being used in extending the stope tracks. It is essential for convenient shovelling that the stope tracks are right up to the face—that is, never more than six feet away from it.

When the footwall tracks are blasted, that portion of the face between the tracks above and below the one being extended is not drilled on, so that no mixing of ore and waste occurs.

The stoping is done by ordinary wet jackhammers, two boys operating a machine; the driller, lying down, operates the machine with his feet (Fig. 1), and the helper, who assists at starting, stops and starts the machine and changes the drills; no learner or any other boys are used, except one drill carrier. Three boys form a crew.

All the holes are marked off with chalk by the miner, using a hole director, and the boys have little difficulty in following the marks.

The broken ore is handled by scrapers, which have an 8-foot handle and a blade 14 in. by 8 in. (Fig. 2). The boys scrape the rock from the stopes into the stope tracks, where it is shovelled into the stope cars and then dumped into the original winze and raise, and from there shovelled into the skip. The cleaning-out boys required for a 400-foot stope are two scraper boys, two car boys, two skip boys, one winch boy, and a barring boy (who is the boss boy). The stope benches are carried at least 8 feet apart. Two holes are required to 'bring' the face, and these are drilled and blasted the same day. The average burden is two feet, and the average depth of hole four feet. Each stoper has two

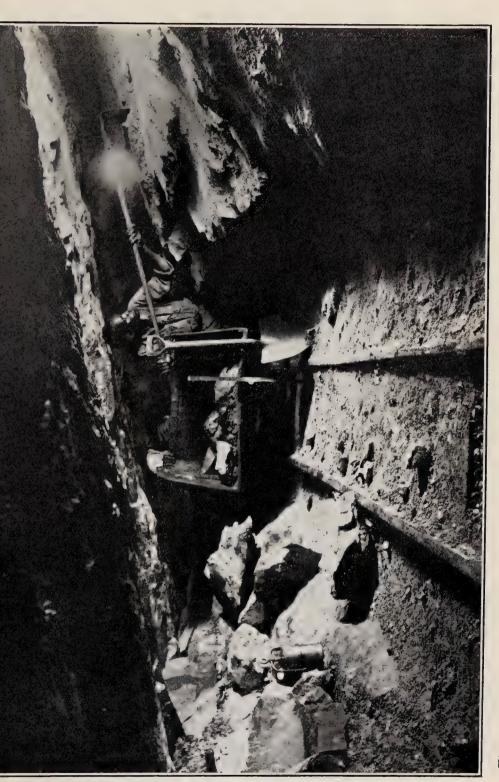


Figure 2.—Shovelling boys scraping broken ore from the stope into stope truck; also stope car and short-handled shovel can be seen. A 24-inch rule is against stope truck.

machines drilling on the overlapping shift, and drills and blasts from 80 to 90 holes per day, using about nine holes, or 36 feet, per fathom broken; and he breaks about ten fathoms, or 60 tons, per day at 24-inch stoping widths.

Form of Contract for Stopes:

The contractors are paid the usual guarantee of 15 shillings per shift, plus a bonus based on four items:

(1) Stoping width: this keeps the stoping width down.

(2) Fathoms per machine shift: keeps them at it.

- (3) Feet drilled per fathom broken: stops presentation of feet to drillers.
- (4) Fathoms per case of explosives: saves explosives. *Method of Support:*
- (1) Four-foot, 4-inch lagging pigsties packed with waste.

(2) Waste packs.

(3) Concrete pillers, 12 in. by 12 in., with 2-inch headboards and a wedge.

General Remarks:

The footwall, which everywhere is shale with a strong parting, is fairly regular. The hanging is good without partings, so the hanging has to be cut, which is an advantage in keeping a regular stoping width.

The lowest average stoping width of a normal stope, that is one of 400 feet 'back', is 16 inches; but occasional pots as low as 10 inches occur, and stretches of around 14 nches are frequent, which indicates that a stoping width of from 14 in. to 18 in. is possible by this method.

Getting about becomes easy, and the narrow width soon loses its novelty.

Crepe rubber elbow and seat pads are used.

The biggest man we have working at the mine weighs 220 lb., and is over 40 years of age; and the tallest man is 6 ft. $7\frac{1}{4}$ in. This will give an idea of the ease of moving about in the narrow stopes.

Costs:

The cost per fathom broken remained the same. Items such as explosives and supervision increased slightly, whereas timbering and shovelling decreased slightly. The cost per ton broken increased, of course, in proportion to the reduction of the stoping width.



Figure 3.—Shows stope being converted from stoping to resuing. The 24-inch rule shows the 18-inch undercut of waste; the 6-inch band of reef shows above the rule.

C.—RESUING STOPING

In starting a new stope, that is from a winze or raise, the waste is stoped out under the reef about three feet in, and the broken waste either hoisted to surface or packed in old stopes. The reef is then blasted down systematically: holes are drilled at right angles to the face about two inches above the reef, two feet apart and two feet deep, and blasted with one stick of powder. With our conditions, the reef is brought down cleanly and completely; if the holes are drilled longer and further apart they are apt to 'bull ring' and also leave 'scales' of reef behind. The explosive used is 40 per cent gelignite for both the stoping of waste and bringing down of the reef. After the reef blast, the ore is handled as in stoping, with scrapers and shovels, and then the footwall is carefully scraped, swept, and washed, so that all fines are recovered; no washing or scraping is done after a waste blast, and any reef that has broken away is hand-picked out and heaped together. When the face has advanced sufficiently to require stope tracks, the same lay-out as for stoping is made, and then all further waste is shovelled back from the face and the stope becomes completely filled with waste rock, leaving the stope tracks open, and a passage down the face about six to ten feet wide.

In mining the waste out, the width is kept low enough so that the broken waste will just completely fill the stoped-out area. Where the 'reef' is very narrow, more waste than this must be broken to give the necessary working width, and the surplus waste is trammed away.

The waste rock, when broken, requires about 50 per cent more space than in the solid, and a minimum convenient mined out width is 18 inches, the same as in stoping.

As there is a strong parting between the reef and the shale, the 'under' waste breaks fairly cleanly from the reef, and, for purposes of tonnage calculations, it is assumed that 3 inches of 'under' waste is taken as 'ore' in the form of fines.

The average waste 'carried' above the reef is five inches. Therefore, taking the reef width at a minimum of four inches, the theoretical minimum resued width of ore is 12 inches. In actual practice, half that width has been achieved where the reef has been four inches or less.

Resuing, instead of stoping, is done where the reef width is below 12 inches. The cost of resuing and stoping is about the same.

D.—CLOSE SORTING-OUT OF WASTE ROCK ON SURFACE

In order to check the surface sorting, and also to find what results could be expected by further screening and sorting the product sent to the mill as fines, the following experiments were carried out:

(1) The ore hoisted was passed over a grizzley with $1\frac{1}{2}$ inches between the bars, the fines going direct to the stamp mill, and the coarse into a washing trommel, and from there to a sorting belt where waste was sorted out to the extent of about 16 per cent of the total ore hoisted. About half a ton (1,020 lb.) of ore was run through the sorting plant in the usual manner, the grizzley bars being approximately $1\frac{1}{2}$ inches apart, and an experimental grizzley with bars $1\frac{1}{4}$ inches apart operating on the fines.

Results from 1½-inch grizzley

Reef Waste to sorting belt	340 lb.	=	33.33	per	cen	t of	total
Waste) to softing beit	200 "	=	19.61	66	44	66	46
Fines and washings	.480 "	=	47.06	66	66	66	66
_							
Total 1	,020 "	=	100.00	66	66	6.6	44
This gives 53 per cent 'coarse' and	1 47 per	ce	nt 'fines	,			

Results from 11/4-inch grizzley

(2) The fines from the $1\frac{1}{4}$ -inch grizzley were then screened, using a 1-inch mesh, and sorted, with the following results:

Results from 1-inch mesh

(3) The fines from the 1-inch mesh were then screened, using a \(\frac{5}{8} \)-inch mesh, and sorted, with the following results:

Results from 5/8-inch mesh

This gives 83 per cent 'coarse' and 17 per cent 'fines'.

Summary:

Screening with $1\frac{1}{2}$ -inch grizzley gives 19.61 per cent of sortable waste.

Screening with $1\frac{1}{4}$ -inch grizzley gives 23.34 per cent of sortable waste.

Screening with 1-inch mesh gives 28.24 per cent of sortable waste.

Screening with 5%-inch mesh gives 37.06 per cent of sortable waste.

Plotting the percentages of 'reef' against 'waste' in the coarse products for each screening gives a straight line, indicating that the percentage of waste increases uniformly between the limits of $1\frac{1}{2}$ -inch grizzley and $\frac{5}{8}$ -inch mesh. This was expected, as the shale waste shatters more readily than the reef or quartzite.

Numerous experiments with 50-lb. samples confirmed the results obtained with the half-ton sample.

Actual Sorting Results:

Prior to the reduction of stoping widths, and with the grizzley bars at 2 inches apart, about 14 per cent of the total ore hoisted was sorted out as waste. The grizzley bars were brought in to $1\frac{1}{2}$ -inch between bars, and then to $1\frac{1}{4}$ -inch, and finally a trommel with 1-inch apertures was installed to deal with the minus product from the grizzley at $1\frac{1}{6}$ -inch.

The following table gives the progressive results of sorting:

Plant with 2 in. between grizzley bars, 14.0 per cent sorted from ore with 66 per cent of waste in it.

Plant with 1½ in. between grizzley bars, 15.8 per cent sorted from ore with 47 per cent of waste in it.

Plant with $1\frac{1}{4}$ in. between grizzley bars, 17.9 per cent sorted from ore with 47 per cent of waste in it.

Plant with 1-in. trommel, 20.3 per cent sorted from ore with 47 per cent of waste in it.

After stoping widths had been reduced, the grizzley bars were brought in to $1\frac{1}{2}$ inches apart, and the following table gives the results obtained:

Waste sorted out

This shows that 56 per cent of the extra sortable waste removed by the $1\frac{1}{4}$ -inch grizzley was actually sorted out, and 50 per cent of the extra sortable waste removed by the 1-inch trommel from the $1\frac{1}{4}$ -inch grizzley minus product was sorted out.

Assuming that the efficiency of sorting the 5%-inch mesh product is 25 per cent, we have the following table:

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Waste sorted with 1\frac{1}{2}-in. grizzley, 15.8 per cent = 80 per cent efficiency " " 1\frac{1}{4} " " 17.9 " = 76 " " " " " " mesh, 20.3 " = 72 " " " " " \frac{5}{8} " " 22.6 " = 61 " "
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As there is an increasing amount of sortable waste screened out as the mesh is reduced, even though there is a drop in the efficiency of sorting the smaller waste, effective extra sorting is obtained, so that it would probably pay to screen down below 1-inch mesh.

The plant has now been re-designed to operate as follows: The ore will pass over a 2-inch grizzley, the 'coarse' going to a washing trommel and then on to a sorting belt;

the 'fines' will feed into a ½-inch trommel, its fines going to the stamp mill and its 'coarse' to a second washing trommel and second sorting belt, so that the 'large' waste will be sorted on one belt and the 'small' waste on another.

The following test rates of sorting were obtained, the sorted waste having the average assay value stated:

Native sorting large waste.—5,343 lb. in one hour, lumps ranging from 25-lb. to ½-lb. Assay value, nil.

Native sorting medium waste.—1,063 lb. in one hour, lumps ranging from 5-lb to ½-lb. Assay value, nil.

Native sorting small waste.—1,156 lb. sorted in one hour, lumps ranging from 2-lb. to one ounce. Value, 0.1 dwt.

The average sorting results for the plant are, $\frac{1}{3}$ ton per boy per hour for 40 boys sorting for 8 hours, and the average assay value of sorted waste is 0.24 dwt. per ton.

SUMMARY

Previous Results:

Reef	hoisted	from	stoping,	12-in.	width	ı ==	26.4	per cent	of ore l	noisted
Waste	44	4.6	66	28-in.	44	==	61.6	6.6	66	66
Reef	46 .	44	development,	12-in.	64	=	1.7	6.	66	6.6
Waste	66	4.6	661	72-in.	64	=	10.3	4.6	4.6	6.6
						-				

Total ore hoisted...... 100.0

Present Results:

Reef hoisted from stoping and resuing, 16-in. width = 58.7 per cent of ore hoisted.

Waste hoisted from stoping and resuing, 8-in. width = 29.3 per cent of ore hoisted.

Reef hoisted from development, 12-in. width = 3.0 per cent of ore hoisted. Waste hoisted from development, 36-in. width = 9.0 per cent of ore hoisted.

Combined Results:

	Per cent Waste in ore hoisted	Per cent Waste sorted out on surface
Previous		20.5 per cent 53.0 "

Costs, etc.:

	Per ton milled	Per ton sorted
Before close sorting		4s. 7d.
After close sorting	12d.	4s. 11d.

An accurate comparison of costs is always a difficult matter—so many factors change. The supervision was about doubled, and this naturally increased the efficiency of items like machine stoping and shovelling, as well as the general efficiency. The following table gives the total gain in profit, due to the combined effects of all the methods introduced. It is difficult to apportion to each method its effect. Both the tonnage milled and the reef value remained practically the same before and after the introduction of the elimination methods.

	Cost per ton	Profit per ton
	milled	milled
Before	38s. 0d.	15s. 7d.
After	44s. 3d.	32s. 6d.

INTRODUCTORY REMARKS AND DISCUSSION

MR. G. J. V. CLARENCE (South Africa): Mr. Chairman and gentlemen, Mr. Quince asked me if I would read his paper for him before this meeting, but as none of the papers are being read I will just make a few remarks on it.

The name of the paper is perhaps a little bit misleading. It signifies the application of clean mining to the Sub Nigel. The Sub Nigel has the narrowest ore body of any mine on the Witwatersrand, and has been working for some years with an average stoping width of forty inches, so that it would appear to be the least promising mine on the Rand to apply clean mining methods to. This width has now been cut down to an average of twenty-five inches for the whole mine, notwithstanding that there are one or two stopes where the reef is forty-five inches wide. There are quite a number of stopes that are being worked on the extraordinarily low width of sixteen to eighteen inches. A great many mining engineers did not believe these figures, but lately a number of them have crawled into some of these places, where they could not even turn over, and have become convinced.

The four methods that Mr. Quince adopted for elimination of barren rock are: (1) resuing development; (2) stoping at narrow widths; (3) resuing stoping; (4) close sorting out of waste rock on surface.

(1) Although resuing stoping is, of course, a very well known practice, the rudiments of which we were all taught at college. I confess that resuing development was something quite new to me, and it may be new to a great many of those present. The method adopted is that the whole round in the development end is drilled over, but, at first, only the holes in the waste below the reef are blasted. This is cleaned out and the reef is then dropped comparatively clean. The average reef width is about twelve inches, and usually about six inches of waste below and six inches above would have to be mixed with it, and we thus take out two feet of ore as against seven feet in the ordinary way. Instead of the average value of the pay development rock being in the neighbourhood of five or six pennyweights, it is by this method increased to one ounce. Development rock in one period amounted to from forty to fifty per cent of the total ore mined, so that it made an enormous difference to the average grade of the ore sent to the mill. whether this source of supply was valued at six pennyweights or twenty. This had a very marked effect on profits. reduction, without any other factors coming in at all, the profits were increased from £10,000 per month to over £20,000 per month, the tonnage remaining at about 10,000 tons per month.

Since that time the tonnage has increased to 20,000 and a new set of factors has arisen; but by these methods of eliminating the barren rock, the profits were doubled. That, of course, is a thing that certainly appeals to directors and I am sure it also appeals to shareholders.

I might mention another point: I believe this is the shortest paper being read before the Congress, and I hope a great many members will read it. There are thirteen pages and only the first nine are really devoted to solid reading, the last few pages being calculations. I think it would be well worth while for those who are interested in clean mining to study this paper.

The mines I have so far seen in Canada have not offered much opportunity for the application of narrow stoping methods. At Kirkland Lake the width of the ore-body would vary from about five feet to forty, and in the places I went through they are very fortunate in having the gold distributed over wide widths yielding a plentiful supply of ore. The mine I actually visited was the Lake Shore, and it is certainly very fortunate in that respect. The Hollinger mine also has a very wide ore-body. But there may be cases of Canadian mines with narrow ore-bodies where these methods would be applicable, and they are worth very careful study.

The Sub Nigel is becoming quite a place of pilgrimage for mining men on the Rand to see how these methods can be put

into practice.

(2) The second heading is "Stoping at Narrow Widths". That is, pure straight stoping carried down usually to between eighteen and twenty inches. Resuing methods are not adopted there.

- (3) If the reef narrows below twelve inches down to two or three inches, resuing is adopted. In this case the footwall is undermined only for a very short distance ahead. About three feet is taken out first and packed back. Then the reef is dropped comparatively clean and trammed out. The reef is cleaned from the stopes about twice a week, whereas the waste is taken out three or four times a week. When the reef is taken out of a stope it is done very thoroughly indeed by the use of hand scrapers, brushes, and washing. If sometimes a little waste is taken out with the reef, this cannot be avoided. The waste is not so carefully removed, but is merely lashed back in the ordinary way, and any that is left is mixed with the reef when it is blasted.
- (4) "Close Sorting Out of Waste on Surface". It is an extraordinary thing that there should be much waste left for surface sorting when mining is so cleanly done. The sorting on this mine was always rated fairly high, averaging from eighteen to twenty per cent, and it is now sixteen per cent. According to all theories of sorting taught at college, the impossible is now being accomplished.

The first improvement made was to reduce the spacing of the grizzleys, on which the ore from the mine is tipped,

from two inches to one and a quarter inches. Both over and under size from the grizzley are passed through separate perforated trommels where they are washed very thoroughly, far more thoroughly than is usual. The oversize go to one sorting belt and the undersize to a second belt. Only the underflow from the washing trommels go direct to the mill.

I have here some samples of pieces of waste that were sorted out on the second belt. It is really astounding that these very small pieces can be sorted out economically.

Another important point is that the narrow stoping was only made possible by the advent of the jack-hammer, and also by the special method of working the jack-hammer. The jack-hammer is worked by the feet and not by the hands. I expect that is known to a great many of you but perhaps not to all. The old method of using the jack-hammer where it was necessary to sit up to hold it, meant that a minimum of thirty inches was required. Now the natives who work these drills lie on their backs and hold the hammer up to the face with their feet. The drill is operated by a second native.

The first problem met with in narrow stoping was getting away the broken rock. Overhead lashing was quite impossible, as there was no way for the natives to swing a shovel, and entirely new methods had to be adopted. Stope tracks to the face were put in as close as every twenty feet, the foot-wall being blasted up to get a height of about three feet six inches. The natives then take out the broken rock by scraping it down with long-handled flat scrapers shaped like a rake.

I hope I have not missed any of the very important points. These are the ones Mr. Quince asked me particularly to stress. If any thoughts occur to any members present and they would like to ask questions, I shall be very glad to answer them if I can do so.

LIEUT.-GEN. SIR WILLIAM FURSE (England): Every member in this party knows by now what a fraud I am. I know nothing about minerals or mining, but before this very interesting paper is discussed by the experts I want to ask the speaker one question.

I can well understand this new method. It seems to me very interesting from an economic point of view, being most satisfactory both to the directors and the shareholders. Would this gentleman who has made the position so clear to us just answer this one question: Is it equally pleasant and agreeable to the miner? (Laughter)

I have no personal knowledge of a technical nature about this, but humanity is the same whether in the air or down in the submarine and on the surface of the earth. Possibly it is the colour of their flesh that makes them take to this kindly. Perhaps you would be kind enough to answer me.

MR. G. J. V. CLARENCE: Mr. Quince has something in his paper if I can find it. He says: "Getting about becomes easy and the narrow width soon loses its novelty. The biggest man we have working at the mine weighs 220 pounds and is over forty years of age. The tallest man is six feet seven and one-quarter inches. This will give an idea of the ease of moving about in the narrow stopes."

As a matter of fact, the work of the actual drilling is done by natives, or we probably would not have got down to the extraordinary widths we have. The native doesn't mind working in an uncomfortable position. The miners get about quite easily. It is surprising how they can get about. The stope tracks run to the face every twenty or twenty-five feet. They are therefore never very long in an uncomfortable position, as they come out on the track and have a little rest and then go down to the next stage. It is really surprising how quickly the shift bosses and men do get about. They get very used to it. It is a peculiar movement. You move on your elbow and on your side and are protected by rubber pads. It is not particularly tiring.

MR. H. WALKER (England): This method of mining is very like the method adopted in mining coal in Somerset. On the point General Furse just mentioned, let me say that the Great Vein in Somerset is only two feet two inches thick; the Little Vein is ten inches thick and the props supporting the roof are thirteen inches long. The mineral is worked; boys pull the coal out in little carts along this thirteen inches of space.

I went to Somerset in 1902 after reading an old book showing this method of mining, which was said to have been abandoned; but I was horrified to find that it was still going on. The boys had a rope around their waist and a chain passed between their legs and they travelled along on hands and feet. In 1914 there was an Inquiry as to whether this method of haulage affected the health of the boys or not. The report was to the effect that it did not. A further Inquiry is being held at the present time.

LIEUT.-GEN. SIR WILLIAM FURSE: May I, if it is in order. Sir, thank these two gentlemen for giving us this information, I told you I was ignorant. These are very important points, and as an old soldier I am delighted to hear that you can do physical exercises underground!

MR. E. C. Andrews (Australia): Sir William told us that humanity is to be found in the airplane and down in the mine, and I can tell him of an experience in coal mining in Australia.

We have there a very narrow coal seam, in a very fine summer resort known as Katoomba, and some of the 'boys' come in from other occupations, such as shearing sheep in the interior. They usually spend their cheques not too wisely and many of them have found themselves stranded, without funds, in the Katoomba area. It was customary for some of them to earn a little money at the Katoomba mines. In this work they had to lie on their sides without pads for the hips, and thus painfully chop at the coal. I may tell you that as soon as they earned a few pounds sterling they left that district as fast as they could. We are beginning, however, to develop a conscience in these matters, and I think it is considered by the government in New South Wales as not the proper thing to ask white men to do such work.

SIR ALBERT E. KITSON (Gold Coast): I should like to thank Sir William Furse for bringing up that point. The first time I saw that picture it seemed to me a great shame to inflict that kind of work on anyone. I am especially pleased to learn from Mr. Clarence that the native miners have no objection to it. I can understand this by knowing how African natives worked for alluvial gold years ago in thousands of small round pits, up to 30 feet in depth, on stream flats in the Gold Coast.

Many of those pits are only four to eight feet deep, with the auriferous portion confined to the lowest one foot or eighteen inches. Large numbers of them are connected at the bottom and the whole of the payable gravel has been extracted. To do this the men must have worked on their faces or sides. It is interesting to note that at some such places there has been no sinking of the surface. The reason is not apparent.

I remember going through the workings on one of the small seams in the Korumburra coal-field in Victoria, Australia. During a strike some of the men voluntarily started to work one of the thin seams. When examining the workings I had, in two or three places, to progress in a prone position. I think that mining under those conditions should be allowed only when undertaken voluntarily. I am glad to know that in the Sub Nigel mine the natives like to work lying on their backs and using their feet on the drills.

MR. G. J. V. CLARENCE: I would like to arise at this stage on a point that has been overlooked. First of all, twelve or eighteen inches below the reef is blasted to a depth of three feet and packed behind in the stope, and then the twelve inches of reef is blasted down, so they have from twenty-four to thirty inches for the average resuing width, although the plain stoping width is from sixteen to twenty inches. The idea is to convert the whole mine to resuing. It is the more efficient method. Of course it takes time, and probably a year or more may elapse before the whole mine is converted to this method.

Professor S. J. Truscott (England): This particular reef is one which occurs at the contact of overlying quartz and a soft shale underneath. Its actual thickness generally, I suppose, is an average of about twelve inches. The shale beneath has no value at all.

Many years ago I was in that district and had the opportunity of seeing just how the Nigel mine was worked. In the Nigel mine they worked almost entirely by resuing. They advanced in the slate and having advanced a sufficient distance they dropped the reef and in that way they mined extraordinarily cleanly, so cleanly indeed that they obtained a very handsome profit with relatively small reduction equipment.

From a battery of twenty-five stamps working cleanly from this relatively narrow pebble bed they achieved extraordinary financial success. But with the lapse of time and with the taking over of this Nigel mine by a large financial group working on the mines near Johannesburg, there came the desire to work on a large scale, to work almost wholesale, and in doing that they lost the advantage of clean mining. In the Nigel district they have never recovered that advantage until this recent introduction of clean mining now at the Sub Nigel.

The great advantage which the introducer mentioned of clean mining arises from the fact, of course, that the same reduction, by treating less waste, is able to treat more ore,

and annual profits rise.

I was particularly struck, in reading this paper, with the fact that the waste sorted out assayed so low; the assay in one case was *nil*, in the second case *nil*, and in the third case one-tenth of a pennyweight. Those are very low figures for waste sorted out of broken ore. Generally the waste will assay more. Probably at the Sub Nigel mine some of the waste at least is the poor shale broken from below the reef.

Professor R. K. Warner (United States): It may be of interest for me to mention some of the thin coal seams that are mined in the Northern Pennsylvania anthracite field across the line. Seams as thin as twenty-four inches on a rather flat pitch are mined in rooms three hundred feet long. The coal is handled by a power scraper to the gangway below. Gangways are made of working height by taking out the floor. In considering the general problem of thin seam mining on the Rand, I wonder if sufficient thought has been given to the use of power drawn scrapers. Scraper practice has been developed in the Michigan copper and iron fields and other parts of the United States to such a high degree of efficiency that it might be worthy of consideration even where labour prices are as low as they are on the Rand.

PROFESSOR G. A. WATERMEYER (South Africa): I would just like to mention one of the oldest mines as being one of these low stoping width mines. I refer to the Mansfeld copper shale mines. I think, if you refer to Agricola, you will

find he was a mining engineer in 1550 and he mentioned the low widths there.

I have been at Mansfeld and also in the Sub Nigel, and I think there is very little to choose between them. The Mansfeld ore is only six inches in width. The men lie on their sides and use picks to pick out the ore. Boys come into the stopes with wooden trays, put the ore in the trays, and creep out through narrow ways with the trays tied to their ankles. That was the method about thirty years ago but I have a very good recollection of it.

On the Sub Nigel, I think one of the main factors which makes narrow stoping such a success is that, from the manager down to the native, every man in the mine seems to be imbued with the idea of making this a success and they leave no loophole for failure.

We should record our thanks to Mr. Quince for his contribution of the paper, and also to Mr. Clarence for the very clear way in which he laid it before us.

THE CHAIRMAN: Before proceeding with the next paper, I will ask Mr. Clarence whether he has anything to add in reply to the discussion on this paper.

MR. G. J. V. CLARENCE: The only point I would like to mention is in answer to Professor Truscott's inquiry with regard to the low value of residues. I, personally, inquired on that point, and the only explanation I can offer is that the waste is washed so thoroughly before it is sorted that all adhering small portions of valuable reef are completely removed before sorting. The waste itself has no value, and if it is clean when sorted it would naturally return a value of *nil*. Samples were taken on three different occasions, assays of which proved the waste as sorted to be valueless.

CONTRIBUTED DISCUSSION

MR. GEORGE A. DENNY (Rhodesia): Mr. Quince's paper will be read by mining engineers all over the world with unusual interest.

The title of the paper—so at least it seems to me—rather obscures its main purpose, which is to show the enormous improvements made in the whole position of the Sub-Nigel

mine since Mr. Quince had the courage to adapt a mining system to the conditions of his mine, and to discard one which had vainly endeavoured to successfully adapt the mine to a preconceived method.

It is true that resuing and clean reef-mining are eliminators of waste rock, but the term 'eliminate' has been so closely associated, in the past, with processes of 'sorting', that the very important work which Mr. Quince has done at the Sub-

Nigel is not envisaged by the title.

Apart from some details which I shall attempt to further elucidate, I do not propose to offer any criticism of the minutia of the methods of working at the Sub-Nigel. Considering that the mine is in the midst of a change-over in both its development and mining systems, the efficiency of the plans in operations, as reflected by the commercial outcome, must have already reached a high order.

Still higher efficiencies may be expected with concurrent favourable effects upon the Company's financial results, when the whole area of the mine will be worked by resuing, and

clean reef-mining.

Resuing Development.—Under this heading Mr. Quince gives certain data concerning the dimensions of working widths and values of reef, etc., which offer some ground for comment.

By resuing 24 in. of mixed waste and reef (that is, 12 in. of reef and 12 in. of contiguous waste) in workings averaging 9 ft. wide and 7 ft. high and a reef dip of 12 in., Mr. Quince says that he "eliminates about 80 per cent of the waste rock from the pay development". The figure I get is 70 per cent. My calculation shows:

For every ft. of advance, $1\frac{1}{2}$ tons of resued reef, at 12 cu. ft. per ton. """"""" waste (12.75 cu.ft. per ton).

Working on these figures, I agree with Mr. Quince's value of 6 dwt. per ton when the whole face of reef and waste is blasted down simultaneously.

Mr. Quince says "the only disadvantage of the method is that the rate of advance is halved". In some circumstances where development was behind or faces limited, that might, of course, prove to be a serious disadvantage. There are

important compensating advantages, the bearing of which might be shown by making a comparison of the commercial results to be obtained from an average development face, as described by Mr. Quince on page 699, opposing the figures of:

- (a) Resued ore from the development face.
- (b) Unresued ore, mixed ore and waste, from unresued development face.

The basis of cost is supposititious. Doubtless Mr. Quince will substitute the correct figures in his reply.

Underground Data:

Schedule A

Method	Dimensions of working	Width broken	Tons per ft. ad- vanced	Assay value of ore	Cost of development per ft. advanced
Resued	9 ft. wide	9 by 2	1.5	20d.	80/- ft.
Unresued	7 ft. high	9 by 7	5.0	6d.	

Cost of milling, general and administrative charges, etc.— These are taken to be ten shilling per ton (hoisting, tramming, etc., included in underground data):

Total cost against ore from 1 ft. of advance:

	Mining	Milling, etc.	Total
Resued	80/-	15/-	95/-
Unresued	80/-	50/-	130/

Gold recovered and profit (or loss) per ft. of advance:

	Tons treated	Gold in 1 ft. of advance	Assay value of residue after treat- ment, per ton	Total gold recov- ered, dwt.	Value of gold recov- ered	Total cost against gold recov- ered	Profit or loss per ft. dev.	Profit or loss per oz. of gold re- covered
Resued	1.5 5.0	30 dwt. 30 dwt.	0.5 dwt. 0.5 dwt.	29.5 27.5	123/ - 115/-	95/- 130/-	,	19/- Profit 11/- Loss

On a basis of the average monthly development advance at the mine, say 1,200 ft., a profit of £1,680 would be won by milling the resued development, as against a loss of £900 if the unresued development rock were milled.

The obvious inference to be drawn from the figures in the schedule above is that by resuing a narrow reef an otherwise

unprofitable body becomes profitable.

Resuing Stoping.—Under this heading Mr. Quince gives details of the methods employed in the operation of stoping by resuing. He states that in "actual practice" where the reef is 4 inches wide or less, the total width of mill-ore mined has not exceeded 6 inches.

He also states (1) that when the reef is below 12 in. in width it is resued; (2) that the cost of resuing and stoping is about the same.

In order that the economic limits and benefits of resuing may be brought home to mining engineers in general, I have set down some leading questions on the subject in the hope that Mr. Quince might supply the desired information.

1. What are the comparative detailed costs involved in the production of a ton of mill ore, and separately, of an ounce of gold from a block of ore in the Sub-Nigel mine, of the following, or any other usual, dimensions:

1,500 ft. on the strike and $\left.\right\}$ Reef width 4 in. 350 ft. on the dip $\left.\right\}$ " value 60 dwt.

In the one case the block is to be considered as having been developed and mined by resuing 6 inches of reef, and, in the other case, by the ordinary methods of development and stoping, which have been used in the past at the Sub-Nigel mine.

The basis of milling for both systems might be taken at a rate of 20,000 tons per month.

The necessary adjustments for 'sorting' would, of course, be made on the unresued ore.

2. In what period would the total gold content of the block assumed have been won and what would be the comparative cost per ounce of gold under the respective mining systems?

3. What is the comparative cost per ounce of gold won under the head of 'overhead' or 'standing' charges in the two cases?

Pending Mr. Quince's reply to the preceding questions, it may not be amiss to forestall his remarks—and incidentally his criticisms—by some prior comments.

The figures deal with a block of ore calculated to assay

40 dwt. per ton by resuing. 7.5 dwt. (raised by sorting) by ordinary mining.

The loss of gold in the waste residues from the milling of the entire block would be:

21,9 116,6	900 to 540	ons o	f resued unresued	ore at	0.7	7 dwt	nces "
The "	total "	gold "	recovered "	from "	the "	e resued block	nces

The cost figure is one that it is hoped Mr. Quince will supply. In the meantime for the purpose of bringing out the advantages of the resuing system, we can proceed with figures which are believed to approximate the actual.

Resuing Stoping

Cost of working, per ton Mining Development Pumping Hoisting	Resued ore 18/- 30/-	Unresued ore 12/6 10/6
Transport Breaking Sorting Stamping Tubing Cyaniding	7/6	7/6
General	7/-	7/-
Total, per ton	62/6	37/6
Total cost Total value of gold produced Total profit or loss on gold recovered Total cost per ounce recovered	£68,400 £182,800 £113,600 P. 31.8/-	£218,000 £174,200 £43,800 L. 106/-

It is hardly necessary to observe that the increased cost per ounce of gold recovered under the ordinary unresued system of mining is due to the fact that the expenditure incurred in the handling treatment of an additional (approximately) 95,000 tons of barren waste rock, not only adds nothing to the revenue but reduces it by an amount of two shillings per ton appropriated from the ore incorporated and lost in the discarded residual treatment products.

A further very appreciable figure might be added to the deductions under the heading of 'unresued ore', if an amount equal to the interest and redemption of the amount expended on the plant necessary to treat the barren rock were to be taken into account, as it should be.

As Mr. Ouince has modestly refrained from setting out the spectacular improvement in the fortunes of the Sub-Nigel Company, which has been effected by him and his staff, with the collaboration and active interest of Mr. W. E. Turveythe Company's consulting engineer—and as this improvement in the Company's financial position affords the most striking testimony to the advantages of resued and cleanly mined ore over unresued mixed-mined ore, in the special conditions of the Sub-Nigel mine and district, I have prepared the subjoined schedule from the published official records of the Company.

The figures for 1923 represent a year prior to the initiation of the clean-mining era. Those for 1927—good as they are represent only the early transition period from the old system to the new. When the latter will have completely supplanted the former the results should be still more favourable for the Company.

Schedule showing improvement effected in the fortunes of the Sub-Nigel Company by clean-mining and resuing

		1923	1927
Tons mil	led	108,000	182,000
Yield	Dwt. per ton Value per ton (standard gold value) Total ounces Total value	12,686 53/6 69,016 £290,830	18,552 78/7 168,818 £715,440

Schedule showing improvement effected in the fortunes of the Sub-Nigel Company by clean-mining and resuing—*Continued*.

		1923	1927
Tons mille	ed,	108,000	182,000
Cost	Dwt. per ton (excluding residue losses) Value per ton. Total ounces. Per ounce saved. Total value.	10 42/2 54,000 66/- £229,540	10 (approx.) 41/8 89,500 45/- £379,200
Profit	Per ton milled (Standard gold value) Per ounce saved Total profit	11/4 18/- £60,600	36/11 39/- £335,840
Ore reserves	Tons in blocked-out reserves. Inch-dwt. value of reserves. Ounces in blocked-out reserves. Profit in blocked-out reserves.	259,000 493 156,735 £146,725	1,013,000 500 1,013,000 £1,009,770

The chief points of interest in the schedule are:

1. That, although the tonnage milled was increased by only 1.7 times, the profit earned increased by $5\frac{1}{2}$ times;

2. The ore reserves increased in tonnage by approximately 4 times;

3. The profit in ore reserves increased by approximately 7 times;

4. The cost per ounce of gold produced dropped by 21/- per ounce;

5. The profit per ounce of gold produced advanced by 21/-per ounce;

6. The value of the ore proper was almost identical in the respective years.

Having achieved such a signal success as is outlined above by a change of mining methods, is it possible to hope that Mr. Quince may place the public and the mining profession under a further obligation by dethroning the present system of basing the results of gold mining operations on the yield and cost per ton of ore milled and substituting it by the only logical term, *viz*: of the cost per ounce of gold produced?

This is the procedure which I, and perhaps many other engineers, have advocated for many years past, and we believe

the argument for the abolition of the cost per ton basis is unanswerable, for reasons which, amongst others, may be set out as follows:

The cost per ton basis is a legacy from coal and iron mining practice, in which it is a perfectly rational scheme, since the mined material—making all allowances for inclusions of foreign matter—is overwhelmingly the material sold for revenue purposes. A statement of results based on the cost per ton of coal or iron mined, and sold, is therefore understandable by everybody.

In gold mining, on the contrary, the overwhelming mass of material mined is a waste product. In the 'ore' mined from narrow veins, the ratio of gold to waste rock may easily be as low as one in 300,000 to 400,000 parts.

Thus, whereas in the case of coal and iron a high percentage of the total material mined represents a saleable product, in that of gold only an infinitesimal part of the material mined is sold.

To know how much it may have cost to produce a ton of worthless material associated with the gold provides no gauge by which the efficiency of gold production may be measured, because by increasing stope widths the tonnage of waste rock may be indefinitely increased without augmenting the gold yield by one iota. A logical realization of the highest efficiency of working a gold mine on the cost per ton principle would, in fact, be achieved by increasing the stoped widths by three or four times that normally worked. If this were done, the cost per ton of rock mined might drop sensationally, but the cost per ounce of gold produced—the only saleable material—might also have risen to well above the selling price of gold, say 85 shillings per ounce.

In the instance of the Sub-Nigel, it would be possible for Mr. Quince to increase the stoped widths in the mine to an average of 60 inches. This would result in reducing the working cost per ton from 42 shillings to possibly 25 shillings, but the cost per ounce of gold produced would be so greatly increased that some stopes now working at, say, 20 inches wide and producing gold at a cost of perhaps 45 shillings per ounce, would become unpayable, because the increase of the stope

width, whilst decreasing the cost per ton, would put the cost of the gold produced from a ton at more than its selling value.

Seeing, therefore, that a figure representing an efficient production of a ton of ore might, and frequently does, represent an inefficient figure on the only saleable constituent of the ton, there are most valid reasons why the ton basis of cost should be cast out, and the ounce of gold basis be substituted.

The ounce of gold basis has the merit of disclosing directly:

- (a) The profit realized, either per ounce of gold, or that from the total gold produced in any period, because gold has a fixed value. A 'ton' may—and often does—represent no gold or other revenue value whatsoever.
- (b) Any increase in the cost of gold production. (The cost per ton can, and often does, give a false impression in this regard).
- (c) The total ounces of gold in the ore reserves, and the profit per ounce, or total, represented.
- (d) The cost per ounce basis makes clean mining the chief object of the working policy. (The cost per ton, on the other hand, puts a premium on waste rock mining).

Indirectly, the cost per ounce basis would control:

- (a) Expenditure on milling plant and equipment generally, since no cost would be lightly incurred which would show an increase to produce an ounce of gold. (Heavy cost is often incurred to decrease the cost per ton, though, in fact, increasing the cost per ounce with no compensating advantage).
- (b) The mining of unnecessary widths of waste rock, outside the minimum working space required for the economic extraction of the reef, since increased tonnage of waste rock mined increases the cost per ounce of gold produced, inter alia, in the following ways:
 - 1. By employing the milling plant to reduce non-revenueproducing waste rock.
 - By reducing the time element in the cost of production, and thus saving very appreciably on the 'overhead' cost.

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I have mentioned the benefits which must accrue by the adoption of the cost per ounce basis in gold mining, not, of course, to spur Mr. Quince on (as he is as well aware as I am of the necessity for the change), but perhaps to invoke further discussion on the matter from Canadian, as well as South African, engineers.

METHOD OF MINING AT WABANA

By J. B. GILLIATT*

(St. John's, Newfoundland, Meeting, September 14th, 1927.)

The ore beds at Wabana occur in a sedimentary formation underlying Conception bay. This formation is made up of beds of shale and sandstone of varying thicknesses. stratification is well defined and the beds persistent. formation is in contact with the older pre-Cambrian rocks at Manuels and at Brigus, near the head of the bay, and dips northerly at about 8 degrees. These stratified rocks appear on three islands in the bay. Below the ore zone on Bell island, there is a thickness of about 4,000 feet of strata measured vertically. The ore zone itself is several hundred feet in thickness and contains three workable beds of hematite ore, all of which have been extensively worked. The Upper bed outcrops on the island near the north shore, is cut off by the cliff, outcrops again in the bay a short distance from the shore, and then dips under the bay. Sufficient cover for mining is attained about a mile off shore. The thickness of the ore bed here ranges from five feet to nine feet. Two other workable beds occur below the Upper bed, namely, the Scotia bed and the Dominion bed. Each of these beds outcrops on Bell island and dips under the bay. Between the Upper bed and the Scotia bed there is a thickness of 58 feet of intervening strata, and between the Scotia bed and the Dominion bed, 240 feet. Between and below these beds many others exist, but all are too thin for mining purposes.

Following the purchase of the mining area on Bell island in 1893 by the Nova Scotia Steel and Coal Company, the mine was called Wabana. A period of aggressive development followed, so that by 1895 the new mine had a shipping pier and a double-tracked tramway connecting the pier with

^{*}Resident Engineer, Dominion Iron & Steel Company, Ltd., Wabana Mines, Newfoundland.

the mine. This tramway was two miles long and was built for conveying the ore from the mine to the shipping pier. Power for drilling was supplied by two 50-horse-power upright boilers set up near the outcrop of the Dominion bed in the locality now known as No. 2 mine. These boilers also supplied power for operating the tramway. Mining started on an exposure of ten feet of ore along the outcrop. The light overburden was stripped off and carried away. Rail connection was made with the new tramway. Benches of ore were cut down and loaded in cars. The rock was hand-picked from the cars during the loading and the men who did the work were called 'rock smelters'. The loaded cars were pulled



Figure 2.—Dominion Pier Loading Plant.

by horses to a bottom on the tramway, where they were attached to the cable by a lead grip and conveyed to steamers at the shipping pier. The empties were returned to the mine by attaching them to the return cable on the empty-car track.

The first cargo of ore mined at Wabana was taken to Halifax and hauled by rail to Ferona, Pictou county, where it was smelted. Later shipments went direct to Pictou. In the next year, 1896, the first contract was filled for a shipment of 50,000 tons to Sparrow's Point, Baltimore. Other contracts were obtained, and the bed was developed easterly and westerly along the outcrop. Northerly on the dip of the ore bed the

open-cut method of mining stopped at the line of heavy cover, beyond which the cost of mining was prohibitive.

After acquiring extensive submarine holdings the Scotia Company sold to the Dominion Iron and Steel Company that portion of the Dominion bed within the land area, and also $2\frac{1}{2}$ square miles of submarine area adjacent to the north shore of the island. By a separate sale the tramway and shipping pier also became the property of the Dominion Company. The Scotia Company built a new pier and tramway further west and started operations on the outcrop of the Scotia bed. After the exhaustion of the easy ore along the outcrop, slopes were started and driven through to the shore



Figure 3.—No. 6 Deck-head.

line. As a considerable tonnage of good ore was indicated, a mining plant was installed. This consisted of a deck-head with crusher and belt house, a battery of boilers, a hoist, and a compressor of about ten drills capacity.

The light overburden was favourable for a high degree of extraction and a large tonnage was obtained, but the land area of this bed was limited and, before its exhaustion, the Scotia Company was looking to its submarine holdings for a future supply.

During the development of the Scotia bed, the Dominion Company was exploiting the Lower bed, which was much thicker than the other beds, ranging from eight to twenty feet over the workable area, and covering a much larger area on land than the other two beds. A period of steady development followed for the Dominion Company. Large areas were stripped of overburden. After the removal of the ore, slopes were started and the land areas developed by the room-and-pillar method.

With many years' supply within the land area, and with three square miles of submarine adjacent to the shore untouched, the Dominion Company was in a comfortable position. Not so with the Scotia Company. The land area of the Upper bed was small and the Scotia bed was nearing depletion. The ore possibilities off-shore appealed to the Scotia management and, accordingly, arrangements were made to drive exploration slopes through the ore bed of the Dominion area to reach the newly acquired holdings. New slopes, now No. 6 slopes, were started in 1905 and completed in 1909. After driving in the Scotia bed for upwards of a mile it was found, by diamond drilling, that thicker ore lay in the bed 240 feet below. Accordingly, the driving grade was changed and the slopes entered the Dominion bed a mile and a half off-shore. The Scotia Company was now in its own areas; a complete mining plant was installed, and No. 3 mine opened up. In order to avoid entrance to this mine through the Dominion ore bed, new slopes were required. These were driven as a tunnel through rock below the Dominion bed. Two headings were driven from the mine towards the island to meet corresponding headings driven from a point near the centre of the island. The horizontal distance from the ore pocket in the mine to the deck was 12,000 feet; the vertical distance, 1,600 feet. The excavation work was completed in 1918 and these slopes have since been in continuous operation. Main levels have advanced easterly in this mine for a distance of 4,000 feet and westerly about 2,500 feet, opening up a large area from which to draw ore.

The steady production of the Dominion Company gradually depleted the open-cut work, and slopes were driven. No. 1 was started at the extreme eastern end of the outcrop, and the land area mined out. Nos. 2 and 3 slopes were started near the centre of the outcrop and the land area of this section was also mined out. Over the Dominion

bed at No. 2 there was sufficient cover at the shore line to continue these slopes under the bay. They are now half a mile off shore, opening up the section now known as No. 2 mine. At the western end of the outcrop, open-cut methods continued until 1920, but before the exhaustion of the ore by this method slopes were driven under the heavier cover and continued beyond the shore line, opening up No. 4 mine.

Progress on the surface kept pace with development underground. Electricity generated at a central power plant replaced steam. The slopes were equipped with electric hoists and the current was carried into the mines, supplying power for lighting, pumping, and driving machinery.

As the occurrence of ore is similar in all the mines there is but little variety in the method of development. But little exploration work is done. The diamond drill explores faults and determines the throw in advance of the workings. It is also used in drilling between beds to find out what lies above or below the working bed. The ore sections of these cores are saved. Thus valuable information is secured.

The workable portions of the ore beds vary in thickness: the Upper bed from five to eight feet, the Scotia bed from six to ten feet, and the Dominion bed from eight to thirty feet.

The slopes are driven in pairs down the pitch of the bed. Through the fifty feet of pillar left between the slopes. cross-cuts are driven at intervals as required for ventilation. When completed, one slope becomes the haulage-way and is single- or double-tracked to suit the method of hoisting. The miners are lowered in cars before hoisting operations start, and are taken out at the completion of the shift. travelling road safe for use at all times is provided in the other slope, which is sometimes called the 'back deep'. Two 8-inch pipe lines are laid in this slope, one carrying compressed air for the drills, the other conveying the water discharged through the pumps. Telephone and power lines also pass through this entrance. These slopes constitute the main. and usually the only, ventilating flues of the mine. The air passes down one of the slopes and, after circulating throughout the mine, returns to the surface by way of the other slope.

With the exception of No. 3, slopes are all driven on the footwall of the ore bed and the hanging-wall usually forms

the roof. Both partings are good and no timbering is required except where the ground is broken at the crossing of faults. When sufficient depth has been attained to open up a new section, a level is broken off at the bottom and driven nearly along the strike in such a way as to give a slight grade towards the slope. Headways are usually driven parallel to the slope.

The first working headway is established near the slopes. A hoist is provided at the top, and rooms broken off on the side away from the slope. These rooms are driven on the same grade as the levels, to assist in the tramming of the ore to the headways. Cross-cuts are driven at regular intervals between these rooms. The rooms and headways form the main arteries of the mine through which the ore-cars pass to the main slope. Rooms and cross-cuts are driven from 20 to 24 feet wide and are spaced to take an extraction of from 50 to 60 per cent of the ore. The balance of the ore is left in the form of pillars for the support of the roof.

Much of the tramming along the rooms is done by gravity, and the cars are placed on a bottom near the headway. Here they are attached to a cable from the drum at the top of the headway, taken out of the room, and lowered to a siding on the main level at the bottom of a headway. The empty cars are picked up by the headway hoist and placed on the sidings in the rooms, to be refilled by the shovellers. The full cars are passed along the level to the main slope, where they are attached to the cable of the main hoist to be taken to the deck head. Twelve cars shackled together form a main trip. They are of steel box type and hold about 1.7 tons each.

The slopes are equipped with 550 h.p. Vulcan hoists at No. 2 and No. 4 mines, and 850 h.p. at No. 6 mine. These hoists are driven by General Electric Co. slip-ring motors. The slope grade runs about 14 per cent, and 1½-inch cable is used. The hoisting speed is about 1,800 feet, and lowering speed about 2,500 feet, per minute.

No. 3 slope is equipped with an 850 h.p. Fraser and Chalmers steam hoist with 11-foot drum. The slope is 2½ miles long and is double tracked for two 20-ton cars operated as balanced load at a speed of 2,800 feet per minute. The small box-cars used in the mine are passed through a tipple

and the ore dumped into a pocket at the bottom of the main slope. From this pocket the ore runs through pneumatically operated chutes into the 20-ton car and is taken on deck.

The drilling is done during the day shift, and great care is required to obtain the proper tonnage of broken ore for the quantity of powder used. An average of about 1.60 tons of broken ore is obtained per pound of powder used, and about 100 tons of ore are obtained from each drill per day. In drilling a square break, the holes are located in rows over the face to be drilled, so placed that they line up horizontally and vertically.

The two vertical rows nearest the centre are sloped so as to take out a V-shaped cut. The next rows on either side nearest the centre are a little less inclined, and the rib holes are driven so as to square up the break. Denver, Dreadnot, and Sullivan drills are used. The size of the steel used runs from $\frac{7}{8}$ -in. to $\frac{11}{4}$ -in.; the lengths vary from two to ten feet, the two-foot starter using a $\frac{1-15}{16}$ in. bit. The size of the bit is reduced as the longer steel is used; for a ten-foot steel the size of the bit is reduced to $\frac{15}{8}$ -in.

Daily requisitions for powder are sent to the thaw house. Here the powder and exploders are packed and sent to each mine, being lowered with the blasters at the completion of the day shift. The holes are loaded by the blasters and wired to battery stations at a safe distance from the face. The centre is fired first and rib holes last. The natural cleavage of the planes at right angles to each other and to the bedding planes gives three ways of easy fracture to the ore, which breaks up into small prisms easily handled by the shovellers.

All rooms fired during the night are gone over by the face cleaners and made safe before the shovellers enter.

In the early days all loading was done by hand shovellers, and much of the ore is still handled in this way. In the thick ore bed of No. 3 mine three Thew shovels are in operation. These shovels require rooms 18 feet high, and a width of 24 feet is required for the swing of the boom. The shovel operates in the same way as the ordinary steam shovel used in excavation work.

A Myers-Whaley shovel is also used in No. 3 mine. This shovel can operate on 5 feet of ore. It is locally called

a 'pig', as it roots into the muck after the fashion of that animal and throws the ore onto a conveyor, which passes it to a car attached to the rear of the machine.

An average output of 90 tons per day is obtained.

Another mechanical loading device deserves special mention. This is the drag scraper. The machinery consists of two motor-driven drums set on a steel frame on wheels.



Figure 5.—Drag Scraper Bucket.

The scraper is made somewhat after the fashion of a slush scraper. In operation, the frame is set up on a level opposite a headway, which is driven directly up the pitch of the ore bed. The scraper is attached to a cable from the drum. The cable from the other drum is attached to the rear of the scraper and passes through a pulley secured at the top of the headway. This cable pulls the scraper to the top of the headway, where it is filled by the operation of the other cable, which pulls it through the muck at the face. It is then dragged to the bottom of the headway and onto a chute, where it is dumped into a car on the level. The scraper rooms are located in No. 6 mine, are 25 feet wide, and are driven 250 feet up the slope. When one of these places is mucked out, the machine is moved along the track on the level and set up at the next scraper room, and an output of 115 tons per day is attained per machine.

Where the cover runs from 200 to 500 feet, as in Nos. 2, 4 and 6 mines, sixty per cent of the ore is extracted. In No. 3 mine, where the cover runs up to 1,000 feet, an extraction of 50 per cent is taken.

The pillars are unaffected by the burden imposed upon them; roof trouble has not developed and no timbering is necessary. Some time in the future much of the ore left in the pillars will be recovered, but no effort is likely to be made in this respect until all other resources of the mine are exhausted.

Although our operations are now submarine, salt water does not enter the mines. Some fresh water comes from the faults and flows to the pumps at the bottom of the mines. The pumping capacity of No. 3 mine is 500 gallons per minute. The pumping is done in three stages from sumps excavated along the slope. The pumps operate $7\frac{1}{2}$ hours daily, and approximately one ton of water is pumped for each ton of ore mined.

In Nos. 2, 4 and 6 mines the lift is much less, and the pumping is done in one stage. No. 2 pumps pass 200 gals. per minute for 7 hours; No. 4 pumps pass 480 gals. per minute for 3 hours; and No. 6 pumps pass 500 gals. per minute for 3 hours.

To date, Nos. 2 and 4 mines have been ventilated by natural draught assisted by chimneys over air shafts with fires at the bottom. Concrete stoppings have been placed in the cross-cuts between the slopes, and many temporary ones throughout the workings to direct the air current to the working places. Both these mines are now being equipped with fans of 100,000 cubic feet capacity.

A fan of 100,000 cu. ft. of air per minute has recently been installed at No. 6 mine and will soon be in operation. A fan of similar capacity is in continuous operation at No. 3 mine and delivers present requirements of about 51,000 cu. ft. of air per minute. The intake slope is two miles long and has an area of 120 sq. ft. The velocity of the intake slope is about 425 feet per minute. Approximately 3,000 tons of air are forced through the mine daily, or nearly 3 tons of air for each ton of ore hoisted.

Although the mines are practically non-gaseous, a small flow of gas does accompany the water from some of the faults, and although it may occasionally collect in pockets it is generally swept away by the ventilating current. Open lights are in general use throughout the mines.

Our accident record compares favourably with those of other mines. During the month of June last, 11,000 tons of ore were hoisted from all mines for each accident involving lost time to the injured. The rate varies considerably, running from two to six accidents per ten thousand man-days worked.

Over 4,000 tons of ore are mined daily at Wabana. In the winter season this output is stock-piled near the mines, and is transported to the shipping piers during the summer. In the shipping season the daily output goes directly to the pockets at the shipping piers. The present output of the mines is in considerable excess of the requirements of the steel plant at Sydney, N.S. The balance is sold in the markets of the world, Germany and the United States being the chief purchasers. The shipping requirements for this season call for 1,300,000 tons.

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